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Rock Burst Research and the Coeur d'Alene District

By Terry McMahon



UNITED STATES DEPARTMENT OF THE INTERIOR



(United States. Bureau of Mines)

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

ft	foot	in	inch
ft ²	square foot	min	minute
ft·lbf/ft ²	foot pound (force) per square foot	psi	pound (force) per square inch
ft·lbf/in ²	foot pound (force) per square inch	s	second
ft/s	foot per second	yr	year
h	hour		

ROCK BURST RESEARCH AND THE COEUR D' ALENE DISTRICT

By Terry McMahon¹

ABSTRACT

This Bureau of Mines report describes the rock burst problem and measures being taken for its alleviation in the deep underground mines of the Coeur d'Alene Mining District in northern Idaho. The geologic features that contribute to rock bursting in that area are discussed and briefly compared with similar features in other burst-prone mining districts throughout the world. Early investigations to understand and control rock bursting are reviewed, and the mechanics of bursting are discussed in terms of the seismic events that produce them. Sources of strain energy, rock failure mechanisms, and the conditions necessary for the release of seismic energy are covered in detail. Rock burst research in the Coeur d'Alene District is described. Included are discussions of the microseismic method as both a research tool and as a rock burst monitoring system, rock destress blasting and preconditioning methods for rock burst control, and stoping methods currently being developed to reduce rock burst hazards.

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INTRODUCTION

Few hazards are feared by underground miners quite as much as the hazard of rock bursting. Rock bursts, unlike fires, explosions, and malfunctions of mechanical and electrical machinery, are beyond the control of the individual miner. Miners understand and are familiar with the safety hazards involved in the use of modern mining equipment and materials, and know that accidents can be prevented by training and safe work practices. Unfortunately, this is not the case with rock bursting. The causes of rock bursts and effective measures for their control and prevention are, at best, only poorly understood. Furthermore, rock bursts usually occur with little or no warning and thus leave individual miners feeling completely helpless and vulnerable against a phenomenon capable of displacing hundreds or even thousands of tons of rock into their workplace.

While mining activities have been practiced for many millennia, rock bursting has only become a problem during the past 100 yr. This is a direct consequence of the development of high-powered hoisting, pumping, and ventilation systems, and the

use of powerful pneumatic drills and high explosives. These developments have made it possible to extend mine workings to greater depths than ever before, and thus into rock more highly stressed than ever before.

The Coeur d'Alene Mining District is one area where severe ground control problems have developed. The incidence of rock bursting in district mines has grown over the past 40 or 50 yr as the mines have been extended to greater depths. Based on past experience, this problem will only become worse with continued mining because the remaining ore reserves are in veins that go still deeper. Today, with the cost of mining higher than at any time in the past and because metal prices are often depressed, the added cost of rock bursting only serves to further threaten the economic survival of the district.

This Bureau of Mines report has been prepared to provide an overview of the Coeur d'Alene District and the geologic environment that has resulted in its rock burst problem, and to review some of the specific research done by the Bureau to help alleviate the problem.

COEUR D'ALENE MINING DISTRICT

The Coeur d'Alene Mining District, located in the panhandle of northern Idaho, is one of many mining areas discovered in the Western United States during the latter half of the 19th century. Unlike so many other mining districts established during that same period, the Coeur d'Alene District's ore reserves were not exhausted after a few decades of intensive mining. Today, as the district enters its second century of mining, the deep vein ore reserves still extend to unknown depths.

INITIAL DISCOVERIES AND MINING

Discovery of the mineral deposits that started the rush to what would become Idaho's most famous and successful mining region was made in late 1883 by A. J. Prichard on the creek named after him.

In January and February of 1884, the rush was on for placer gold, owing in large part to a brochure entitled "In the Gold Fields of the Coeur d'Alene," published by the Northern Pacific Railroad to stimulate business (1-2).² The town of Murray, located in the northern part of the district, as shown in figure 1, was established within a few months, and by early summer placer mining was under way. Further prospecting led to the discovery of the lode deposits that were the source of the placer gold, and by 1885 several arrastras and stamp mills were in operation to mill the gold-bearing quartz (3).

²Underlined numbers in parentheses refer to items in the list of references at the end of this report.

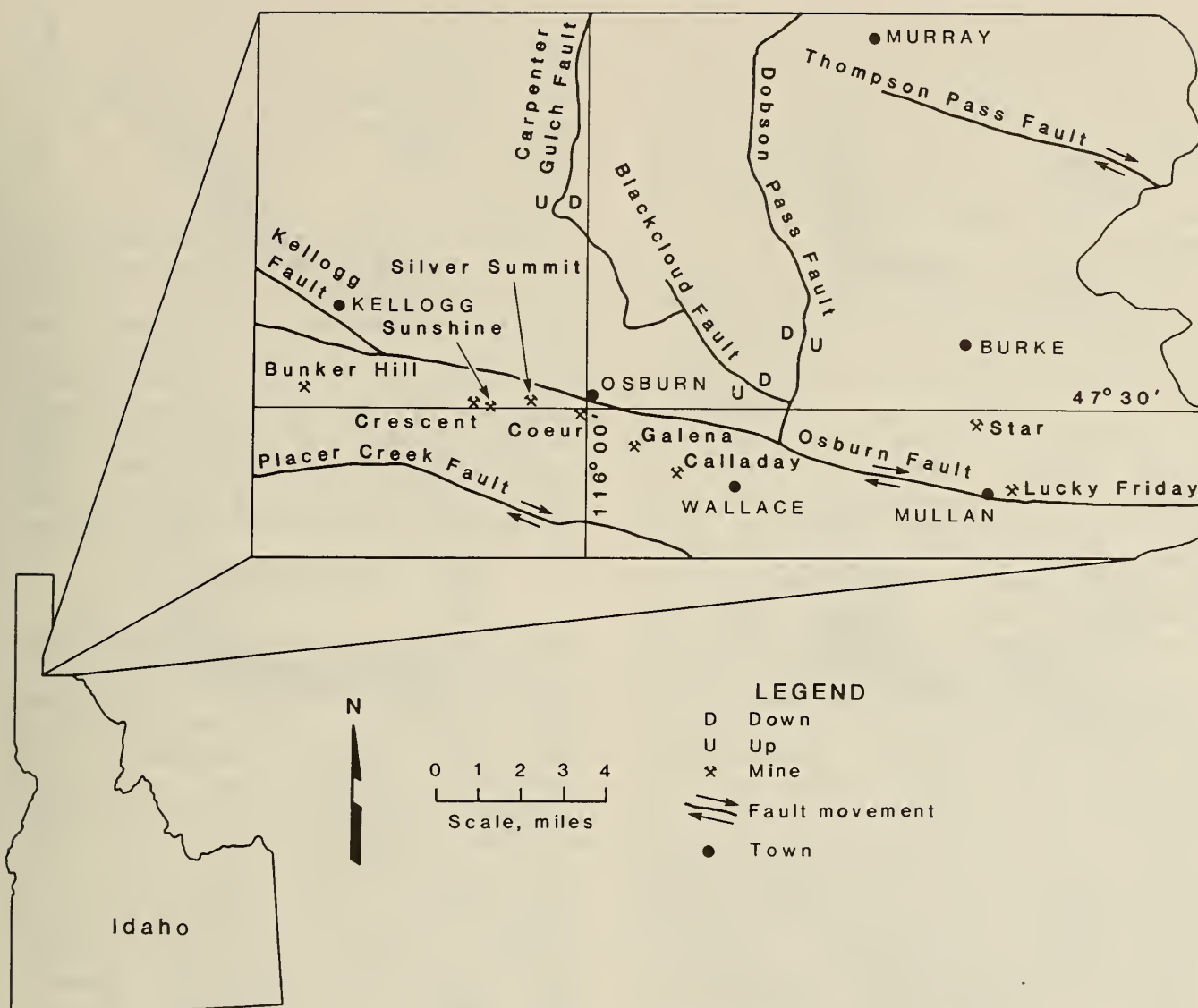


FIGURE 1.—Coeur d'Alene Mining District.

The vein deposits of lead and silver, for which the district would become world famous, were discovered by a disappointed gold prospector, Noah S. Kellogg, who wandered south and west of Murray in late 1885 (4). The discovery was made in Milo Gulch, a few miles south of the town that now bears Kellogg's name. One of the district's most famous mines, the Bunker Hill, was soon in operation, first as a glory hole and then as an underground mine. Within 5 yr, several other major silver-lead deposits had been located in the district and mining operations started. The towns of Wallace, Burke, Mullan, and what would become Kellogg were soon established, and by 1890,

the district was being served by two railroads.

GEOLOGY OF THE DISTRICT

The host rock of the Coeur d'Alene's vein deposits is a regional sequence of Precambrian metasediments known as the Belt Series. This sequence of rock extends from eastern Washington to western Montana and underlies all of northern Idaho. The Belt Series consists of a thick conformable group of generally fine-grained quartzites and argillites (5). The series is divided into six formations that range in thickness from about 1,000 ft to over 12,000 ft. In

descending order from youngest to oldest, the formations are the Striped Peak, the Wallace, the St. Regis, the Revett, the Burke, and the Prichard. Mineral changes observed in all Belt Series rocks indicate that at least a slight regional metamorphism has occurred during periods of both high stress and high temperature.

The geologic structure of the Coeur d'Alene District is extremely complex. Following deposition and lithification of Belt Series sediments, a system of principal folds is believed to have formed during a very early stage of deformation. This folding, probably the result of strong tectonic compression from the northeast and southwest, resulted in a series of synclines and anticlines. This stage of structural evolution is illustrated in figure 2A. As deformation continued, deep-seated stresses may have become reoriented, subjecting the folds to a rotational stress that later became a shearing stress. These stresses caused a bowing and warping of the fold axes into a structural knot and also produced shear zones that would evolve into the major east-west fault zones of the district, as shown in figures 2B and 2C. As shear zones evolved into continuous fault zones through the region, some of the strong, deep-seated tectonic stresses were probably released through the large fault displacements shown in figures 2D and 2E. The Osburn Fault is the most prominent fault of the district and has been mapped almost continuously for a distance of over 100 miles, from near the town of Coeur d'Alene, ID, to a point east of Superior, MT. The Osburn is a right-lateral, strike-slip fault with up to 16 miles lateral displacement and a nearly vertical dip through the district.

Folding and faulting continued through the Larimide Orogeny and was accompanied by a period of monzonite stock intrusions. This was also the period of principal ore deposition, which occurred in narrow, nearly vertical veins often extending to unknown depths (fig. 2F). The veins are generally located in linear bands along fractures and fault zones, and are generally found in the quartzites

of the Belt Series as opposed to the argillites (6). Continued stress and fault displacements have produced the tight folding and complex arrangement of fault offsets observed today both on the surface and in the deep mines of the district.

The high tectonic stresses responsible for most of the district's complex structure have left the area with an unusually high horizontal stress field. Measured horizontal stresses in district mines range from 1.5 to over 2.0 times the vertical stress (7). The vertical stresses encountered at the depths now being mined are often in excess of 6,000 psi. These factors--a hard, brittle rock mass in a very high stress field--produce a very high potential for violent rock failures.

ROCK BURSTS

Rock failures in underground mines may be referred to as either "rock falls" or "rock bursts." Rock falls, as the name implies, refer to falls of fractured, loose rock into mine openings. While potentially dangerous to miners and equipment and costly in terms of cleanup and lost production, rock falls are relatively nonviolent events.

Rock bursts, on the other hand, involve the violent fracturing and explosive displacement of rock into mine openings. While a rock fall is generally a gravity-induced event, a rock burst results from the sudden and violent release of stored strain energy within the rock mass near a mine opening. When released, the strain energy becomes seismic, or shock, energy and travels through the rock as a seismic wave until the surface of an opening is encountered. The seismic energy, if of sufficient magnitude, is capable of fracturing and displacing large volumes of rock. The magnitude of the seismic energy may vary over an extremely wide range, and as a result, the volume of rock displaced by a burst may range from no more than a few fragments to thousands of cubic yards. Small seismic events may be so weak that they are subaudible and thus require microseismic equipment for their detection; they do no damage to underground structures. Other seismic

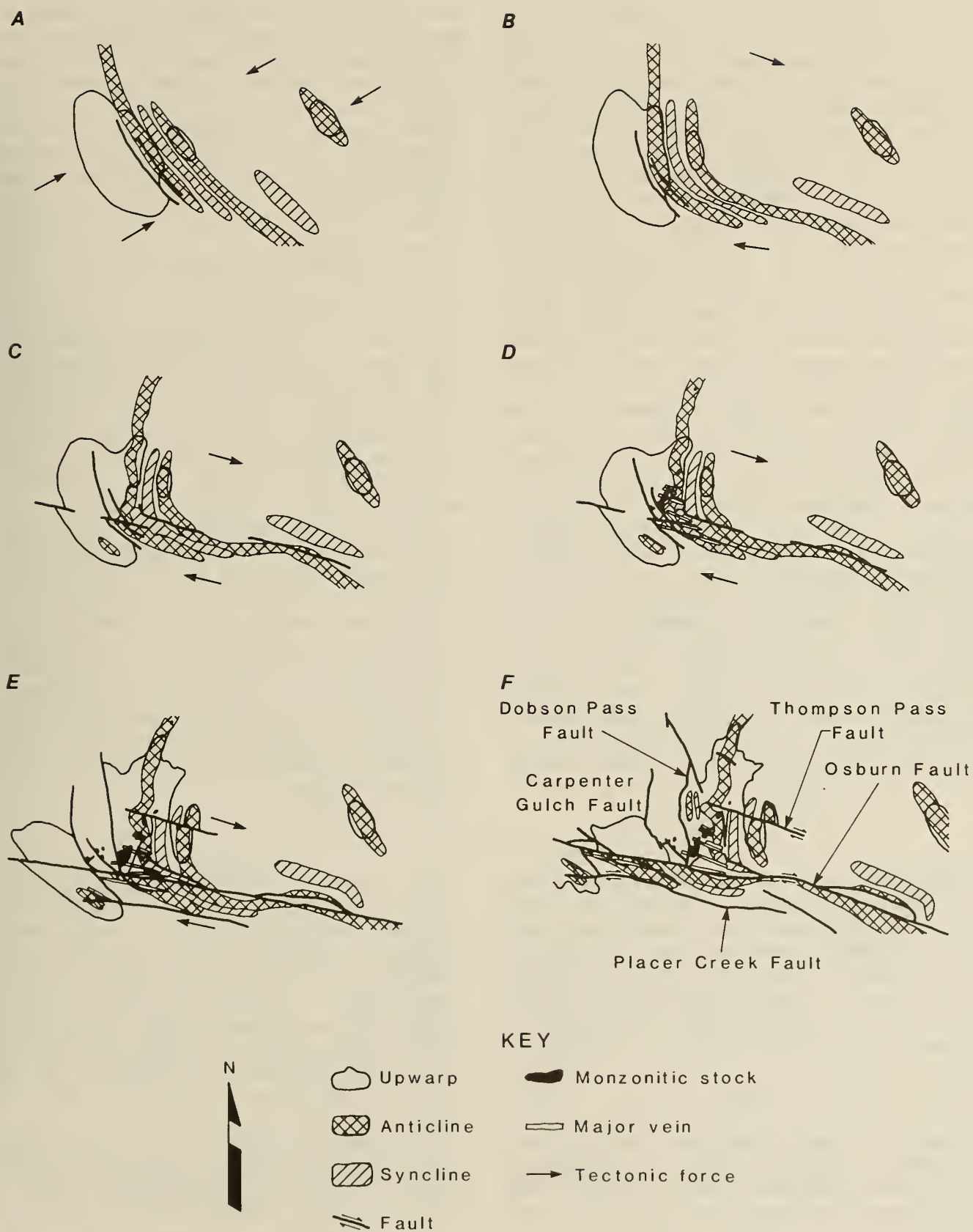


FIGURE 2.—Structural evolution of Coeur d'Alene Mining District, after Hobbs (5).

events, termed "rock noise," may be audible but still do no damage. At the opposite extreme are the large rock bursts, which have been recorded as having magnitudes as large as 2.7 on the Richter scale in the Coeur d'Alene District and much higher in some other mining districts (8). In the case of one Canadian mine, damage from a large rock burst hastened the complete closure of the mine (9).

During the past 50 or so years, with mining depths in Coeur d'Alene mines reaching well below 2,000 ft, rock bursts have become common events. Until closure of the district's larger mines, bursts had become an almost daily event in one mine or another during periods of heavy mining activity (10). Most of these bursts would occur following mine blasting, when personnel were out of the stopes and away from potential burst areas. These bursts would often be relatively minor in magnitude and do little damage. In other cases, a rock burst might be preceded by rock noise in the form of a popping, cracking, or low rumbling sound, which would alert miners (11).

During the early years, miners often referred to rock bursts as "rock blasts" because of their explosive nature or as "air bursts" or "air explosions" because of the resulting airblast rushing through

mine openings. Because of their violent and explosive nature, rock bursts frequently resulted in fatal or very serious injuries to anyone caught by the displaced rock.

The extreme hazard that rock bursts present to Coeur d'Alene District miners has been underscored many times since June 4, 1941. It was then that two miners, William Mir and William Edison, became the first in a long list of rock burst fatalities in district mines (12). The two men were working on the 2900 level of the Sunshine Mine when, at 7:25 p.m., a major rock burst buried the men under tons of wall rock in the stope they had been working. The damage to the stope and amount of displaced rock was so great that rescue teams working continuously required 25 h to recover the first body and almost 3 days to recover the second.

Since those first fatalities, there have been scores of other fatal accidents and serious injuries due to rock bursts in district mines. In addition, rock bursts have done hundreds of millions of dollars of damage to underground mine structures and equipment. As district mines are developed to greater depths, the potential for future rock burst problems becomes greater because of the increased stresses encountered with increased depths.

ROCK BURSTS IN OTHER MINING DISTRICTS

The Coeur d'Alene District is not the only location where rock bursts have become a serious operational problem. The deep gold mines of India and especially the Republic of South Africa have experienced severe rock bursts for many years, as have many other mining districts around the world.

INDIA

The first mining district to experience rock bursts was the Kolar goldfield in Mysore State of south-central India. The first reported burst occurred in the Oorganm Mine in 1898 at a depth of only 960 ft, where gold was being mined from a narrow, steeply dipping vein (13).

The ore vein is a belt of hornblende schist surrounded by granite and gneisses, which have been regionally metamorphosed. The veins range in width from about 3 to 20 ft and have been interrupted by a series of faults, pegmatites, and dikes. These structural features have produced planes of weakness in the rock, which are believed to be the foci of the major rock bursts of the district.

Rock bursts in the Kolar goldfields became a major operational problem as the mines went deeper. The deepest mine is now over 10,500 ft below the surface, and rock bursts with magnitudes of 4.5 to 5.0 on the Richter scale have been recorded. Over the years, rock bursts have been

responsible for numerous fatalities and injuries, heavy damage to mines (including the destruction of otherwise minable ore shoots), and even damage to surface structures located near the mine. In one instance, on October 22, 1937, a major burst was felt 18 miles away and damaged local seismometers to the point that local readings could not be obtained (14).

REPUBLIC OF SOUTH AFRICA

Gold mining on a major scale began in the Republic of South Africa in 1886 with the development of outcrops in the Witwatersrand area of the Central Rand. The gold ore in this district occurs in a narrow conglomerate seam in quartzite and shale. Within 20 yr after the start of mining, when the mining depth was less than 1,000 ft, the first rock bursts or "earth tremors" began to be a problem.

The gold-bearing conglomerate seam or "reef" is part of a sequence of geologically stable Precambrian sediments, which dip from about 45° at the surface to about 25° at the 10,000-ft depths now being mined. The hanging wall consists of a thick sequence of quartzites, and the footwall is a sequence of shales. There are some offsets of the seam due to faults and intrusions by dikes, but the seam is essentially a planar structure. The actual gold-bearing portion of the conglomerate ranges from a few inches to several feet in thickness.

During the early years in the Republic of South Africa, there was a dramatic increase in mining activity as well as associated seismic activity. Seismic records of the Union Observatory indicate only 7 tremors in 1908, but 233 tremors in 1918 when mining was at a depth of only about 2,000 ft. The rock burst fatality rate also rose dramatically during these early years, reaching a peak during the decade of 1926 to 1935. Records for the following 40 yr show a steady decline in overall fatalities but a nearly constant number of fatalities due to rock bursts (15).

CANADA

Deep, hard-rock mining in Canada has a history of rock bursting in districts such as Sudbury, Kirkland Lake, Timmins, and Red Lake, and also in the potash mines of Saskatchewan. Rock bursts were first reported at Sudbury in 1929. Since then, rock bursts at Sudbury, as well as in Canada's other districts, have been responsible for many fatal injuries and enormous damages.

One example of a rock-burst-prone area is the Kirkland Lake District of Ontario where gold is mined from steeply dipping, narrow veins. The host rocks of these veins are Precambrian sediments, tuffs, and igneous intrusives that have been repeatedly folded, faulted, and intruded by dikes (16). This has produced a complex geologic structure similar to that of other burst-prone mining districts around the world.

While several mines have operated in the Kirkland Lake District for many years, the Lake Shore Mine and the Wright-Hargreaves Mine have had more serious rock burst problems. Rock bursts began at the Lake Shore Mine in the early 1930's and shortly thereafter at the Wright-Hargreaves Mine. In 1964, a major burst at the Wright-Hargreaves Mine led to the closing of the entire mine (9).

OTHER DISTRICTS

Other mining areas where rock bursts occur include iron mines in Sweden (17), the copper mines of Poland (18), and various underground mining districts in Europe. Some Australian and South American mining districts have also reported problems with rock bursts.

Underground mining is not the only excavation activity in which rock bursts have been a problem. In highway tunnel projects along western Norway's many fjords, rock bursts have occurred in the form of spalling rock caused by shear-fracturing of the tunnel surfaces. Investigations have revealed that these bursts are caused by both anisotropic

stresses and large horizontal stress components within the steep mountain

sides being tunneled (19).

ROCK BURST MECHANICS

Investigations into the source and cause of rock bursts and attempts to find preventative or control measures began in the Republic of South Africa shortly after bursting had become a serious problem.

EARLY INVESTIGATIONS IN THE REPUBLIC OF SOUTH AFRICA

The work in the Republic of South Africa is divided into two periods: the first beginning about 1908 and extending into the late 1940's, and the second beginning in the early 1950's and continuing to the present. The first phase is known for its empirical approach to understanding the cause of rock bursts and trial-and-error methods of finding workable solutions. The second phase is known for its organized scientific approach to determining root causes and then designing preventative measures.

First Phase

The first phase of rock burst investigation began when the South African Government appointed the Ophirton Earth Tremors Committee in 1908. Following its investigation, this committee concluded that the observed earth tremors were the result of the "shattering of support pillars." Their recommendation was to replace the solid reef pillars being left for roof support with packs of waste material. The Witwatersrand Earth Tremors Committee, appointed in 1915, concluded that the tremors were caused by "rock bursts due to sudden crushing of pillars," but also noted the "fracturing or settling of overlying strata" as an additional cause. This committee also recommended the use of waste packs or other artificial supports as a remedy. A third committee, the Witwatersrand Rock Burst Committee, was appointed in 1924. It classified rock bursts into two categories: "strain bursts," which were small but numerous, and "crush bursts," which were less frequent but much larger (20). This committee also made several

recommendations to help control the bursting problem. Among them were the use of artificial supports, the use of large shaft pillars for protecting mine shafts, and the use of longwall mining. The recommendation to use longwall mining methods indicates that the influence of mine geometry on abutment stress concentrations was at least intuitively understood in the early years. Unfortunately, it was not until many years later that the longwall method was implemented and proved to be an effective rock burst control measure.

The investigative work done during the early years was generally performed by the mining engineers working in the field and thus those who were closest to the rock burst problem. Understandably, their solutions were based on their observations and experiences and their own experimental methods, primarily trial and error. It is not too surprising then that often their recommendations to control bursting were ineffective and occasionally even contradictory. As an example, some engineers advocated the use of solid backfilling for the artificial support of a mined-out area, while others recommended allowing the complete caving of the mined-out area.

By the late 1940's, the South African gold mines were operating at depths greater than 8,000 ft and the rock burst problem was growing worse in both severity and incidence. It then became apparent that available control measures were not effective, and if progress were going to be made, a more organized effort would be required. Furthermore, this effort would require a much more rigorous scientific approach.

Second Phase

In 1949, the second phase of rock burst investigations began with the formation of the Council for Scientific and Industrial Research (CSIR). Then, in 1953, the South African Chamber of Mines assumed full responsibility for research on the rock burst problem. In organizing

its plan of attack, the Chamber of Mines adopted four fundamental approaches to understand and alleviate rock burst phenomena. These were (1) observation and statistical analyses of rock burst occurrences, (2) an intensive laboratory rock testing program, (3) an intensive investigation of the seismicity of the mining areas, and (4) research into the application of mathematical-modeling methods for mine structures.

The objective of the first approach, statistical analyses of rock burst occurrences, was to establish relationships, if possible, between rock burst characteristics and various mining and geologic variables. Some of the important observations resulting from the work were that longwall mining methods reduce bursting, small or acutely shaped pillars produce higher incidences of bursts, burst activity increases significantly near the stope face, and bursting is often associated with certain geologic features such as dikes and faults (21).

The program to test rock under laboratory conditions was initiated to study the behavior of hard, brittle rock under high-stress loading conditions, to gain an insight into the deformation characteristics of such rocks, and to determine their failure mechanisms (22-23). Among the results of this testing was the discovery that hard, brittle rock, when slowly loaded in "stiff" testing machines, tends to exhibit strain-softening following the onset of failure (24-25), rather than fracturing violently as brittle materials normally do when tested in "soft" testing machines.

The investigation into the seismicity of the rock surrounding mines was the first to employ three-dimensional underground seismometer arrays. These permitted the accurate location of seismic events relative to the areas of active mining (26-27). By monitoring seismic activity underground, close to the source, it was found that the number of seismic events was far greater than the number of reported rock bursts. This led to the realization that not all seismic events result in rock bursts, and that the rock mass is much more

seismically active around active mines than had been thought.

Elastic-theory investigations were initiated to determine whether the response of the rock mass surrounding mines could be modeled numerically with models derived from the mathematical theory of elasticity. The displacements predicted by such modeling, when compared with measured displacements in the mines being modeled, indicated that most of the rock mass around mines did behave as an elastic material (28-30). Only the fractured and deformed rock in the immediate vicinity of mine openings, where inelastic behavior was exhibited, could not be modeled as an elastic continuum.

These initial research efforts were only the beginnings of a continuing effort to obtain a basic understanding of rock burst sources and mechanisms and to find effective methods for their control and alleviation. Since these early efforts in the Republic of South Africa, rock burst research has continued not only there but in many other countries with mining districts where bursting has become a problem. The basic research conducted in more recent years has supplied a wealth of knowledge and experience, and has focused on fundamental rock burst questions such as the source of energy, the mechanisms of rock failure that release this energy, and those mining variables and geologic conditions that make a rock burst more or less probable.

DEFINITIONS OF A ROCK BURST

Many individuals and government agencies known for their experience and expertise in rock burst research have offered definitions of a rock burst. Some of these are

A sudden rock failure characterized by the breaking up and expulsion of rock from its surroundings, accompanied by the violent release of energy (31).

Damage to underground workings caused by the uncontrolled disruption of rock associated with a violent release of

energy additional to that derived from falling rock fragments (21).

That phenomenon which occurs when a volume of rock is strained beyond the elastic limit, and the accompanying failure is of such a nature that accumulated energy is released instantaneously (32).

A sudden and violent failure of a large volume of overstressed rock, resulting in the instantaneous release of large amounts of accumulated energy (33).

An instantaneous failure of rock causing an expulsion of material at the surface of an opening or a seismic disturbance to a surface or underground mine (34).

These definitions all point out the key characteristic of a rock burst: the sudden release of energy in the form of violently expelled rock. The seismic energy released during a rock burst may range in magnitude from less than 0.5 to over 5.0, as measured on the Richter scale (35). The amount of failed rock or damage done by a burst will likewise vary over a wide range.

In the Republic of South Africa's deep gold mines, a seismic event is considered a rock burst only when measurable damage is done. Based on past experience, this usually occurs when the seismic energy has a magnitude of 0.5 or more. Other fields, such as civil engineering, may have a definition that reflects higher sensitivity to rock bursting than does that of the mining industry. In such cases, any seismic activity induced by excavation may be considered a rock burst, even when there are no damages or injuries.

SEISMIC ENERGY SOURCES

Since rock bursts are manifestations of seismic energy, it is important to consider (1) the sources of stress that strain a rock mass to produce stored strain energy and (2) the energy balance changes resulting from mining that may release the energy required to cause a rock burst.

Gravitational and Tectonic Stress

In general, there are two natural sources of stress, gravitational and tectonic, that can deform a rock mass and produce stored strain energy. The gravitational stress at a given depth is simply a function of the rock density and depth below the surface. Tectonic stresses, on the other hand, originate from geologic stresses occurring on a regional scale and as such may contribute a larger stress component to the rock mass than the gravitational component. Tectonic stresses are often an important factor in burst-prone mining districts.

Two burst-prone districts where tectonic stresses constitute a major component of the in situ stress field are the Coeur d'Alene District and India's Kolar goldfields. Horizontal stresses in the Coeur d'Alene District range from 1.5 to over 2.0 times the vertical stress (7), and in the Kolar goldfields the ratio ranges from 1.6 to 4.0 (13).

Mining-Induced Stress

The stresses having the greatest influence on the stability of rock surrounding a mine opening are those induced by excavation in the rock. The initial opening and enlargement of mine workings cause disruptions and consequent redistributions of the in situ stress field around the workings. Stress redistribution takes place to compensate for the lost support of the mined rock and results in a concentration of stress about the opening. The total stress is then a combination of in situ and induced stresses. Together, these stresses strain the rock and produce stored strain energy.

ENERGY BALANCE

One of the results of early research was the realization that when an opening is made or enlarged, the balance of stored strain energy is upset (21). As a result of excavation, some energy is released when the newly formed opening surfaces converge slightly to adjust for the lost support of the removed rock.

Early research led to the conclusion that, for an unsupported opening, up to one-half of this released energy would become available as seismic energy. More recently it has been shown that this can only be true if the opening is made in one large step, i.e., removing all the rock at once. Where a mine opening is enlarged in many small steps (small relative to the dimensions of the opening), the released energy can be accounted for almost entirely by the strain energy in the rock being removed by mining. Mining in small steps, as is usually the case, has thus been shown to be a stable activity that does not result in the release of seismic energy (15).

The energy change relationships occurring when a mine opening is enlarged from equilibrium state to another have been rigorously treated by Salamon (36) to determine the influence of the number of steps taken to make the enlargement. When the enlargement is made, energy becomes available from two separate sources. The first is the gravitational or potential energy W from the external and body forces doing work through displacement and deformation of the rock or volumetric convergence of the enlarged opening. The second is the stored strain energy U_m contained in the volume of rock removed to enlarge the opening. The sum of these ($W + U_m$) is then the energy that becomes available as a result of mining and must be expended in some manner.

Some of this energy will be expended by an increase in the concentrated strain energy U_c stored in the rock mass ahead of the opening. If supports or backfill are used, then some of the available energy will be expended by support deformation W_s . If it is assumed that the rock mass is an ideal elastic continuum, then none of the available energy will be dissipated by fracturing or inelastic deformation of the rock. With this simplification, the sum ($U_c + W_s$) is the energy expended by enlargement of the opening.

Since it is obvious that the energy expended cannot be greater than the energy available, and the strain energy U_m removed with the mined rock is not available to do work on the rock around

the opening or deform its supports, the inequality

$$W > (U_c + W_s) \quad (1)$$

must hold, and since $U_m > 0$,

$$(W + U_m) > (U_c + W_s). \quad (2)$$

The difference in the energy available and the energy expended in inequalities 1 and 2 is the excess or released energy W_r . This difference must be expended in some manner, and from inequalities 1 and 2,

$$W_r = (W + U_m) - (U_c + W_s) > 0 \quad (3)$$

and

$$W_r > U_m > 0. \quad (4)$$

This shows that mining activity in an elastic continuum is always accompanied by the release of some energy, which must be expended in some manner. If the mining were done all at once, in one large step that removed all the rock instantaneously, the sudden change would produce oscillations in the rock mass. These oscillations would then be dampened by minor imperfections in the rock, and kinetic energy W_k would be dissipated as equilibrium was reestablished in the rock.

Since no other means of energy dissipation is available, inequality 4 becomes

$$W_r = U_m + W_k, \quad (5)$$

and from equation 3,

$$W_k = W - (U_c + W_s) > 0. \quad (6)$$

Equation 5 indicates that the released energy W_r is independent of the method of energy transfer, whether by the removal of rock U_m or the dissipation of kinetic energy W_k ; however, analytical computations (36-37) have indicated that the relative values of U_m and W_k are heavily dependent on the number of steps taken to enlarge an opening. This is illustrated by the enlargement of both a circular tunnel and a spherical cavity from an

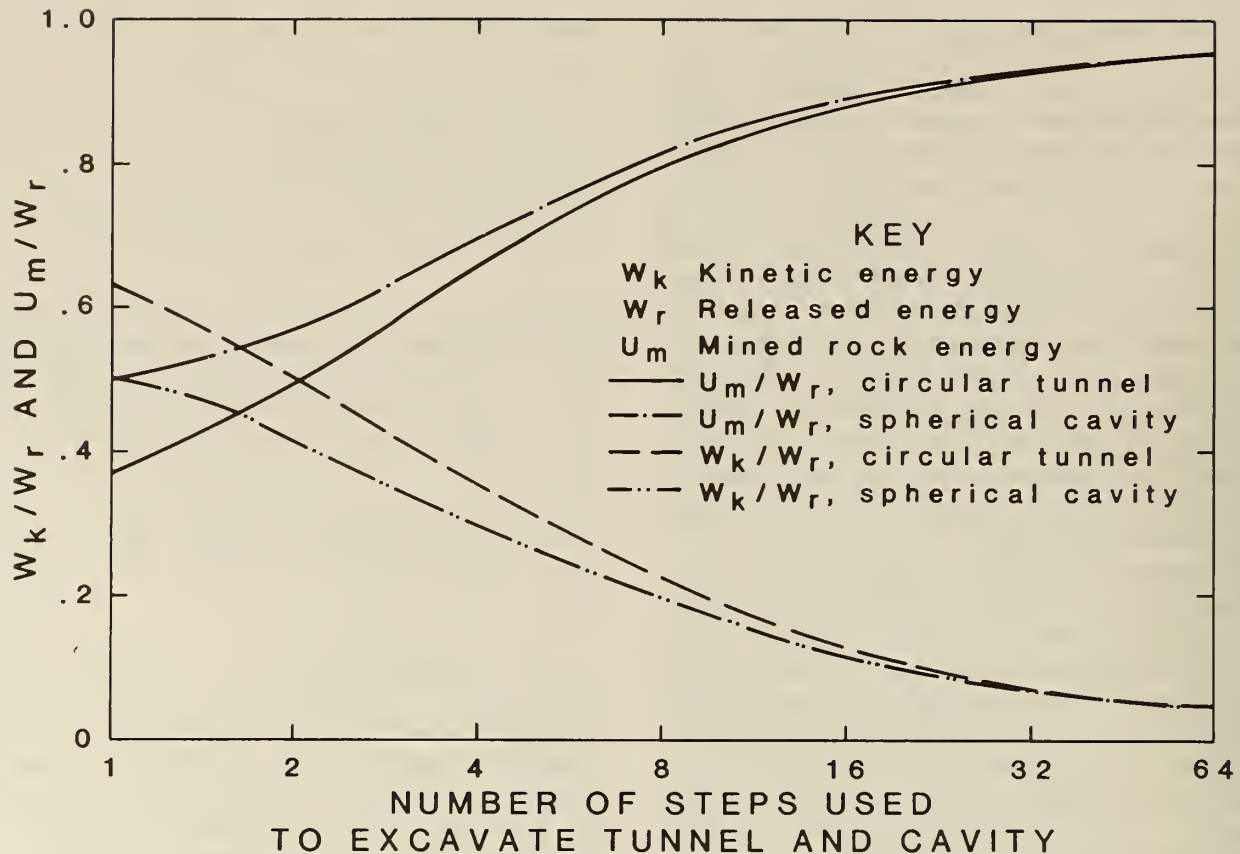


FIGURE 3.—Split of total energy released with increase in number and decrease in size of excavation steps.

initial to a final radius. The number of steps taken to make the enlargement ranges from 1 to 64. In both cases, the energy removed with the mined rock (U_m) increases, while the released kinetic energy W_k decreases as the enlargements are made in progressively more steps.

These results are plotted in figure 3, and as can be seen, when a spherical cavity is enlarged in one step only, U_m and W_k each account for one-half of the released energy. As the number of steps is increased and the size of each step reduced, the energy released as kinetic energy W_k is reduced and eventually approaches 0. The same trend holds true for enlargement of the circular tunnel, in that an increase in the number of excavation steps results in a reduction in the released kinetic energy.

The results of this analysis, when applied to the energy changes due to mining, lead to the following equations:

$$\Delta W = \Delta U_c + \Delta W_s, \Delta W_r = \Delta U_m, \Delta W_k = 0, \quad (7)$$

where the Δ symbol indicates that the mining is done in very small steps.

Based on this work, Salamon reached the following conclusions:

1. The work ΔW done by external and body forces through volumetric displacements is fully expended by the increase in strain energy ΔU_c and deformation of supports ΔW_s , when the change in mine geometry is very small.

2. The energy ΔW_r released by continued mining is accounted for by the stored strain energy ΔU_m in the rock removed by mining.

3. There is virtually no kinetic energy ΔW_k released by mining activity, and thus mining in small steps is a quasi-static, stable process and cannot be the source of seismic energy.

As stated previously, the analysis and conclusions given above pertain to an ideal elastic continuum. As such, the results provide a limiting case when applied to mining. In an actual mining environment, the rock immediately surrounding an opening is usually highly fractured and thus exhibits inelastic behavior. The effect of inelastic behavior is to increase the volumetric displacement, which in turn increases the available energy W and the released energy W_r . While not all of the released energy can then be accounted for by the energy removed with the mined rock (U_m), most of it can be, according to Salamon. Furthermore, the amount of energy that may become seismic energy is much less than previously thought.

ENERGY RELEASE RATE

"Energy release rate" is a term given to the calculated rate at which energy becomes available per unit wall-rock area mined, as a result of mining-induced displacements and stress changes. The basis for the calculation method is the fact that the spatial rate at which energy is released during mining is heavily dependent on mine geometry and the mining sequence taken to reach the final geometry. When a particular stoping sequence induces high stresses and large displacements, the rate of energy release will be high. If the energy released is greater than can be dissipated by deformation and nonviolent fracturing in the rock, the excess energy will be released violently and may produce a rock burst. The calculation method does not assume a threshold value of energy released above which rock bursting will occur but has been empirically related to increases in both the incidence and severity of rock bursting. The energy release rate can be controlled to some degree by the systematic sequencing of mining steps and, as a mine design and analysis tool, has been used extensively for mining tabular ore bodies in the Republic of South Africa where it was developed (21).

The relationship between the density of damaging rock bursts as a function of the rate of energy release, based on mining

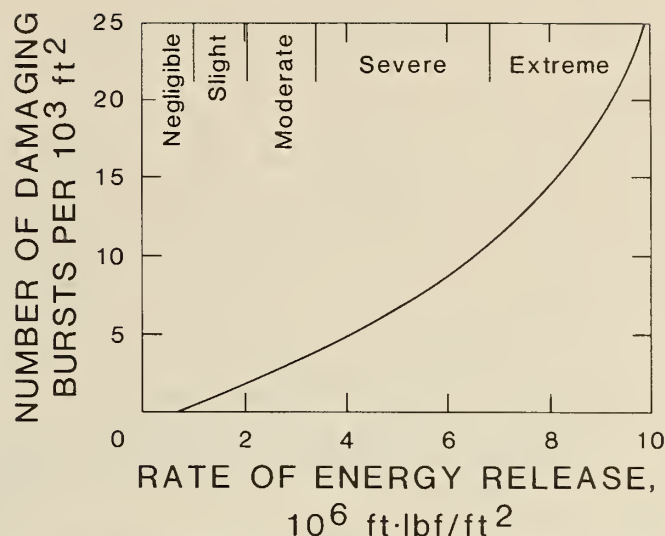


FIGURE 4.—Increase in incidence and severity of rock bursts with increase in energy release rate.

experience in the Republic of South Africa (38), is shown in figure 4. The figure also shows the increase in rock burst severity experienced with increases in energy release.

CONDITIONS FOR A SEISMIC EVENT

In order for stored strain energy to be released as kinetic or seismic energy from a region of highly stressed rock, four conditions are necessary. The rock must be strained to the point of unstable equilibrium; there must be an additional increase in stress to trigger the rock failure; the nature of the failure must be a violent, brittle fracture; and there must be sufficient stored strain energy available to be released as kinetic energy.

Unstable Equilibrium

The condition of unstable equilibrium of a system is said to exist when the system is in equilibrium and its potential energy is at a maximum value. If an attempt is then made to add any additional energy, the equilibrium will be upset and the potential energy of the system will be released as kinetic energy. The pendulum shown in figure 5 is a simple example of this concept. When the pendulum is rotated to the top dead-center

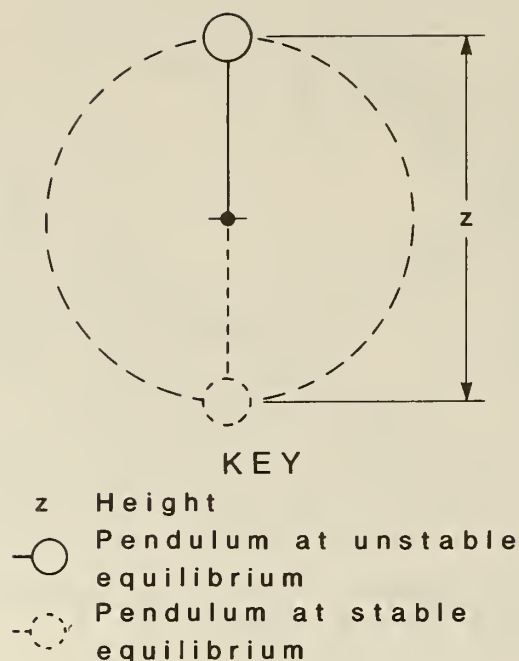


FIGURE 5.—Simple pendulum, illustrating unstable and stable equilibrium.

position as shown, work is done on the system by raising the pendulum through height z , and energy is stored by the virtue of the pendulum's position. The system is then in equilibrium, and any attempt to do additional work on the system will upset the pendulum, causing it to fall and its potential energy to be released as kinetic energy. Once upset, the pendulum will swing back and forth through bottom dead center until the oscillations are dampened and the pendulum comes to rest in a position of stable equilibrium. As will be discussed later, these oscillations are analogous to seismic waves radiating from a seismic event.

The condition of unstable equilibrium in a rock mass means that some point or local region has been strained by natural and induced stresses to the threshold of failure. If just the slightest amount of external work is then done on the rock in an attempt to strain it further, equilibrium will be upset and failure will occur. The stored strain energy in the rock near the point of failure will then be released. If any of this energy should be released as kinetic energy, a seismic event, and possibly a rock burst, will occur.

Triggering Stress Increase

When the condition of unstable equilibrium has been established, an additional stress increase must occur to trigger the rock failure. This stress increase, however slight, may come from the zone of concentrated stress that advances in the rock ahead of excavation, or as is often the case, it may come from the sudden stress change caused by an excavation blast.

The zone of concentrated stress induced in the rock ahead of an opening is shown in figure 6. The concentration of stress produces an inelastic, fractured zone around and ahead of the opening. If the rock is at or near unstable equilibrium ahead of the face, the approaching stress increase may initiate failure. Geologic discontinuities such as dikes or faults can cause an abrupt increase in the stress and trigger failure as the opening approaches the discontinuity.

Violent Failure

Once rock failure is initiated, it must be a violent, brittle type of failure and as such must be accompanied by an abrupt postfailure loss of strength. This condition requires a rock that is strong and hard, and exhibits brittle-material behavior upon failure. A stress-strain curve for a perfectly-brittle rock material is illustrated in figure 7. When such a material reaches its maximum strength and fails, the sudden loss of strength permits a violent displacement of fracture surfaces and a simultaneous drop in stress. Violent displacement of the fracture surfaces will cause the surfaces to recoil and oscillate about their equilibrium position until the oscillations are dampened. This motion is the mechanism responsible for conversion of the stored strain energy into kinetic, or seismic, energy.

Release of Seismic Energy

The violent displacement of fracture surfaces that releases and radiates seismic energy from the point of rock failure requires the availability of excess

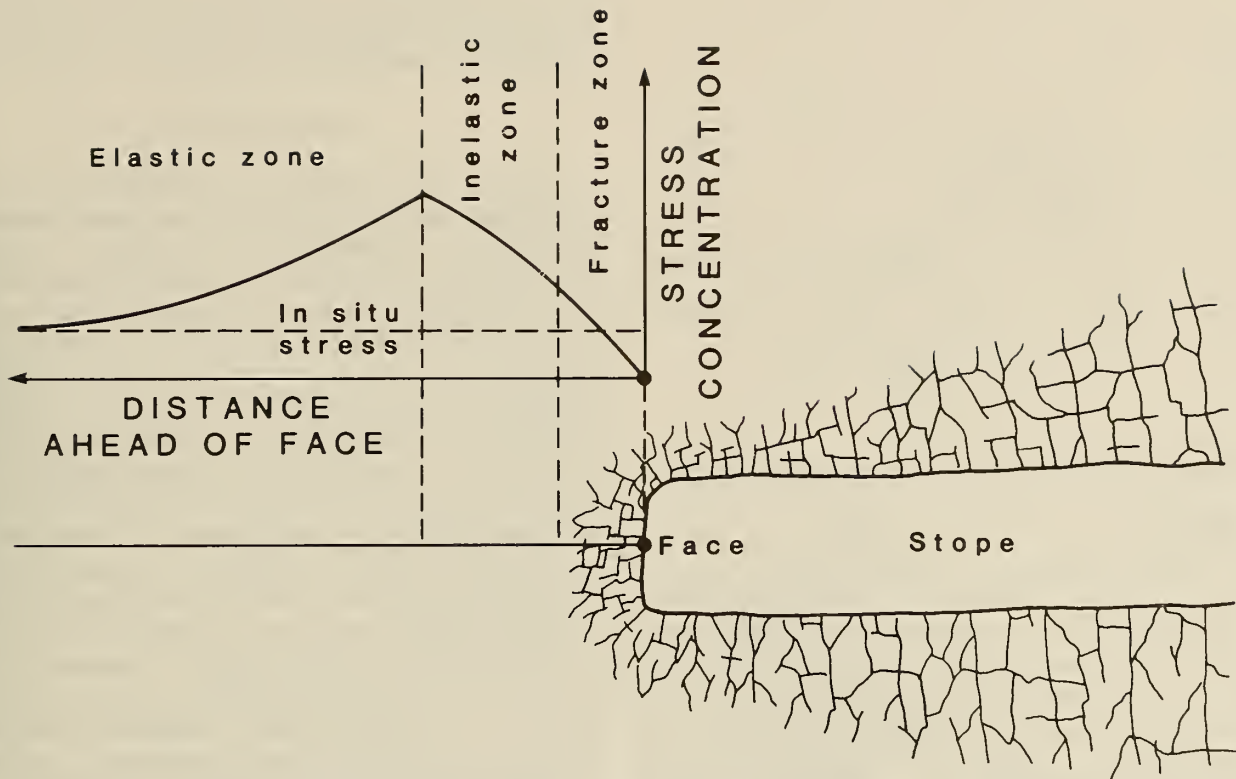


FIGURE 6.—Mining-induced stress concentration ahead of face.

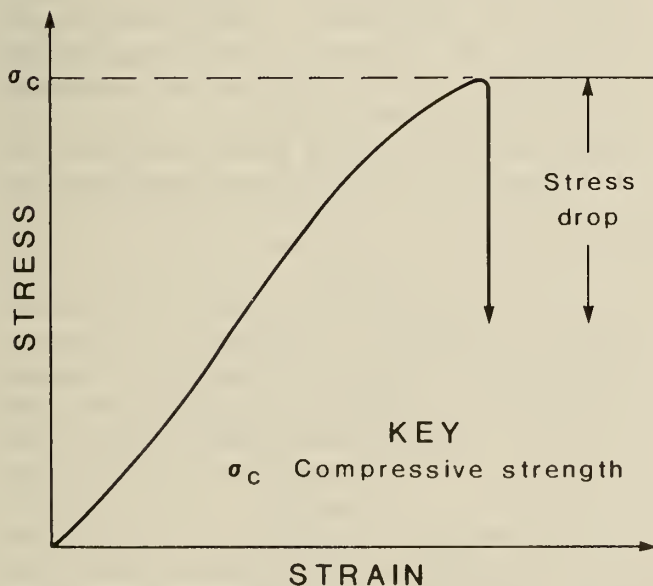


FIGURE 7.—Stress-strain curve for perfectly-brittle material.

strain energy. Excess energy means more energy than can be dissipated by stable, nonviolent rock failure and deformation. Energy availability is related to the

stiffness of the strained rock around the fracture surfaces and the slope of the postfailure stress-strain curve.

The stress-strain curve shown in figure 8A illustrates stability following failure because the slope of the postfailure curve is less than that of the rock stiffness. In this case, there is no excess strain energy available to be released as kinetic energy. The stress-strain curve of the rock in figure 8B illustrates a postfailure curve with the slope greater than that of the rock stiffness. Rock failure under these conditions is intrinsically unstable because increased displacement is accompanied by decreased strength or strain softening, and the surrounding rock will continue to load the fracture surfaces following initiation of failure. Continued loading will result in the violent displacement that releases excess strain energy as kinetic energy, until equilibrium is reached when the falling strength stabilizes at the fractured strength level.

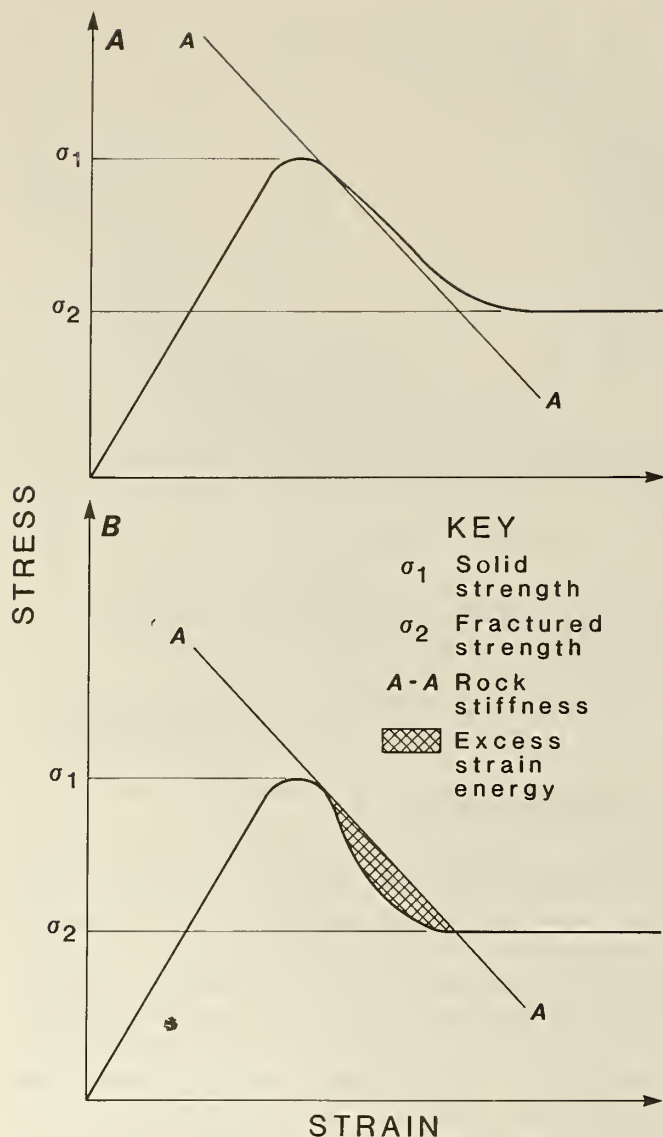


FIGURE 8.—Stress-strain curves. A, Nonviolent failure; B, violent failure due to excess strain energy.

FAILURE MECHANISMS

Rock failure mechanisms involve loading conditions, physical properties of the rock, and material instability characteristics that combine to produce a fracture or failure zone and promote fracture growth once started. The understanding of failure mechanisms is an active and important area of current rock burst research. By establishing rock instability and failure criteria, and then incorporating these into numerical models, it may become possible to analyze

mine geometries and calculate the probability of rock bursting.

Intact Rock

The first efforts to establish failure criteria for brittle rock materials dealt with the initiation of failure and the development of failure planes under shear and normal stress conditions. The Coulomb criterion (35) postulates shear failure as a linear function of material cohesion and angle of internal friction across a failure plane, while the Mohr criterion postulates a functional relationship, which is dependent on the material, between the shear and normal stresses. The Griffith criterion for brittle fracture postulates that failure is initiated with the growth and coalescence of preexisting microfissures within the stressed rock. The microfissures ultimately join together to form the macroscopic failure surface.

While these early efforts concentrated on failure initiation, later research included the behavior of rock in the postfailure portion of the stress-strain curve and the effects of confining stress and elevated temperature on rock response during loading and failure.

Analytical models of postfailure behavior attempt to model failure localization and development within an initially uniform material as an instability in the constitutive description of deformation. When stressed, the material undergoes a normal homogeneous deformation until a bifurcation point is reached. Continued loading then produces localization and development of a planar shear band at some point in the material where non-homogeneous deformation and eventual failure occur. Outside of the shear band, equilibrium is maintained and normal homogeneous deformation continues with continued loading (39).

Another model, which has application to bedded or layered rock, treats the rock failure mechanism as a problem in surface instability and may be considered a special case of the previously described model. This model develops an instability criterion in terms of the

uniaxial and compressive strengths, and applies a bifurcation analysis technique for failure localization (40).

Faulted Rock

The failure mechanism of geologic discontinuities such as faults, joints, and bedding planes is dependent upon the shear and normal stresses acting on the surface and the frictional properties of the surfaces. Shear displacement of the surfaces may be prevented by normal stress, which provides a clamping action through the static coefficient of friction of the surfaces.

If mining-induced stresses should upset the balance of stress on the discontinuity by either increasing the shear stress or reducing the normal stress, then displacement along part of the discontinuity will occur. Whether displacement is stable or unstable will depend upon the stiffness of the rock and the slip response of the discontinuity.

The model of a block on a flat surface, shown in figure 9, illustrates the stability conditions of a discontinuity. The normal force N acts on the block perpendicular to the surface, while the tangential force T acts parallel to the surface through the spring having stiffness k . The system will be stable and the block stationary as long as

$$T < \mu_s N, \quad (8)$$

where μ_s is the static coefficient of friction and $\mu_s N$ is the static friction force. When the tangential force T is increased, or the normal force N is decreased, to the point that

$$T = \mu_s N, \quad (9)$$

the system will be in unstable equilibrium. If there is then the slightest increase in force T , or reduction in force N , the block will begin to slide. Once sliding begins, the kinetic coefficient of friction μ_k , being less than the static coefficient of friction, will allow the block to continue to slide with less force than that required to initiate sliding.

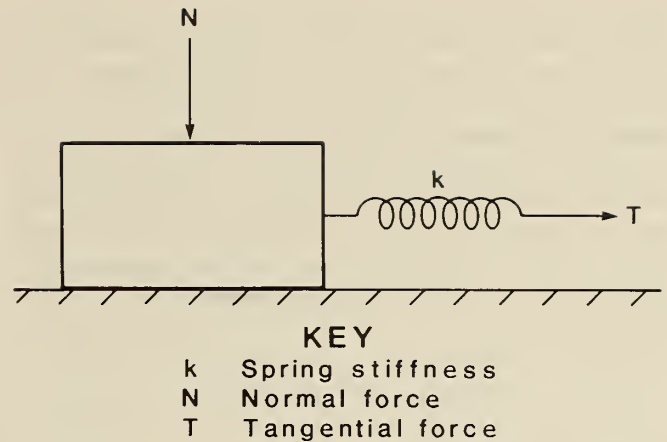


FIGURE 9.—Simple spring-loaded sliding block model.

If the tangential force T required to extend the spring is greater than the force required to maintain sliding on the block-surface interface, the motion of the block will be stable. If, on the other hand, the force required to extend the spring is less than that required to overcome static friction, strain energy will be stored in the spring as the block is loaded by the tangential force T prior to motion. When the static friction is finally overcome and the block begins to move, the stored strain energy will be released on the block, causing an unstable, violent motion.

SOURCE LOCATIONS AND ROCK BURSTS

Whether a rock burst will result from the release of seismic energy will depend on the magnitude of the energy released and the distance between an opening and the point of rock failure. When the radiated seismic energy wave arrives at an opening, the wave is reflected by the surface of the opening. If the magnitude of the seismic energy is sufficient, reflection of the seismic wave will fracture the surface rock and cause it to be violently expelled into the opening.

At the Face

If high stresses in the rock immediately adjacent to an opening's fractured and inelastic zone should initiate a seismic event, the rock that fails will be expelled into the opening with explosive

force. Rock bursts caused by the release of seismic energy at or very near the face are known for their severity in terms of fatalities, injuries, and damages. Ironically, these bursts may be caused by seismic energy releases that are relatively low in magnitude compared with other seismic events. It is the proximity of the opening to the source of energy that causes the severity of the burst.

Ahead of the Face

Another cause of seismic activity responsible for rock bursting is the formation and propagation of shear fractures in the stressed rock ahead of an opening. That these are shear fractures is inferred from the detailed analyses of

microseismic wave forms originating from ahead of openings (41). The locations of these shear failure planes and their orientations with respect to stopes and structural features such as dikes have also been inferred from microseismic data (42). As previously mentioned, features such as dikes may cause abrupt increases in the induced stress zone ahead of an advancing opening and thus promote seismic activity.

In the bedded quartzites of the Republic of South Africa's gold mines, shear-type fractures have been classified according to their inclination to the horizontal and their seismic activity (43). They have also been studied by excavating exploratory drifts and raises into the rock ahead of the face (44). Figure 10 illustrates the rock parting

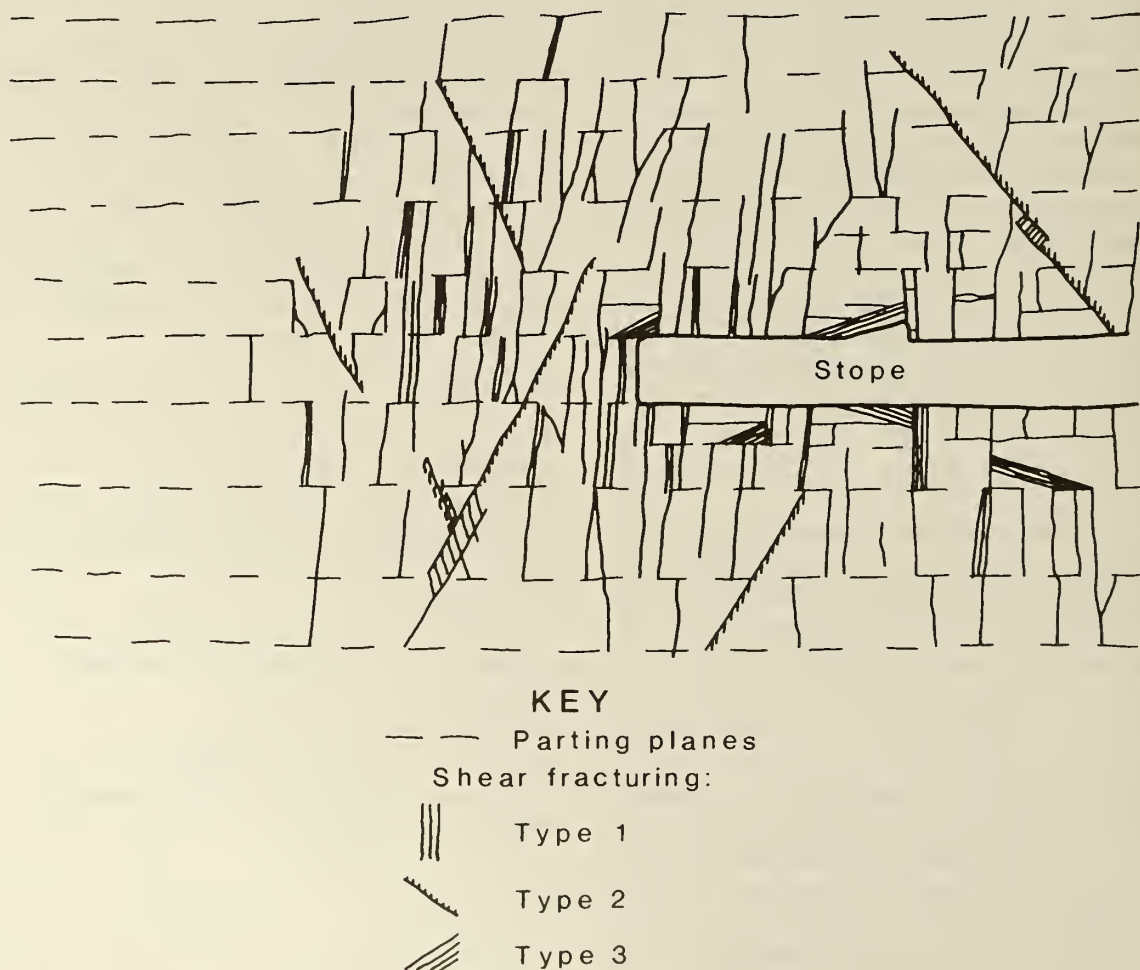


FIGURE 10.—Types of shear fracturing in bedded quartzite ahead of stope. (See text for discussion of types.)

planes and the three types of shear fracture commonly found. Type 1 fractures are the most common, have a near-vertical dip, and are generally planar. Type 2 fractures dip 60° to 75° and may form complex fracture zones as well as conjugate fractures. Type 3 fractures dip 20° to 40° and are usually found at or very near the surfaces of openings.

As in the case of rock bursts from seismic events at or very near the face of an opening, bursts originating from shear fracturing often cause severe damages because the seismic energy is released close to the opening. These fractures are often located within four or five opening widths of the face. Seismic energy magnitudes from these shear fractures have been measured in the range of 0.5 to as high as 3.0 on the Richter scale.

On Faults

Geologic discontinuities such as faults and joints are a third source of seismic

activity and rock bursting in mines. As mentioned in the earlier section on failure mechanisms in faulted rock, mining-induced stress changes may increase the shear stress or reduce the normal stress on a discontinuity and allow surface displacement. If the displacement is an unstable, violent motion, seismic energy will be released.

The Richter magnitude of the seismic energy released by violent discontinuity displacements is often very large, in the range of 3.0 to 5.0, but the damages caused by rock bursts produced by these events is often less than that of bursts caused by other sources. The reason is the greater distance usually found between mine openings and seismic event locations on the discontinuities. These events may be felt simultaneously at several mines in a district and yet cause relatively little or no damage at any of them. When such events do occur close to mine workings, the resulting rock burst damage is catastrophic.

COEUR D'ALENE ROCK BURST RESEARCH

Research into the causes of rock bursts in the Coeur d'Alene District and the development of control and preventative measures began in the early 1940's when the potential seriousness of the problem was first recognized. Since then, the microseismic method has become a refined research tool for rock-mass monitoring and for studying rock failure mechanisms. Rock burst control measures such as destressing, i.e., reducing the level of stress in rock, and rock preconditioning have been developed, and the improvement of stoping methods to control rock bursts continues today.

MICROSEISMIC METHOD

Hard-rock miners have long known that the crackling and popping sounds made by rock in underground openings are related to the stability of the rock. This "rock talk," as miners refer to it, has been taken as a warning of potential rock failure. Any abrupt change in rock talk, especially any increase in noise rate or

amplitude, would prompt miners to vacate the noisy area until the rock either failed or quieted.

Early Development

In 1938, the Bureau initiated research to determine if a relationship existed between the seismic velocity through a loaded support pillar and the state of stress in the pillar. During the experimental phase of this research, while using sensitive seismic equipment, it was discovered that rock under stress emits subaudible or microseismic signals in addition to audible sounds or rock talk. The generation of these microseismic signals in stressed rock implied a relationship between the time rate of the signals and the state of stress in the rock (45). This relationship was soon verified during a seismic investigation of pillars in a deep copper mine in northern Michigan. During the course of the investigation, the ground being monitored became seismically noisy after an initial quiet

period, with the rate of noise increasing for about 15 min before terminating in a rock burst less than 50 ft from the point of observation (46). This unexpected event led to the hypothesis that microseismic activity, if it always preceded a rock burst, might be used to predict or warn of an imminent burst. With this event, development of the microseismic method as a rock burst research tool had its beginning.

The first mention of an investigation into the rock burst problems of the Coeur d'Alene District came in October 1941, only 4 months after the district's first rock burst fatalities at the Sunshine Mine (47). There was optimism that the microseismic method, then in its early stages of development, could provide a solution for the problem. Unfortunately, no systematic method of rock burst prediction or warning was then available, and it was not until the mid-1960's that the microseismic method would be applied in the district when methods for seismic source location became available.

Development of the microseismic method began with both laboratory and field testing. The laboratory testing of rock specimens stressed by testing machines revealed that the microseismic noise rate increased with increasing stress and that, just prior to specimen failure, there was a dramatic increase in the noise rate (48). The source of both the audible and subaudible noise is the microfracturing and microdisplacements that occur as the rock responds to the increasing load. The dramatic noise increase as failure approaches comes from the increase in microfracturing and the growth of fracture surfaces on which failure occurs.

Field testing with microseismic equipment was done by monitoring stressed pillars and rock structures in burst-prone mines (49). The object of this work was to monitor microseismic activity for a period and determine whether repeated patterns of noise increase preceded rock failure, and whether or not these patterns could be recognized. The equipment available at the time did not permit location of the noise sources, and

consequently, the monitored noise included extraneous noise and was too irregular to provide recognizable patterns prior to failure. Also, the monitoring method was highly subjective and heavily dependent upon the skill and experience of the user, which always left an uncertainty in the results.

Seismic Source Location

Research with the microseismic method as a means of determining rock stability continued with fieldwork in underground zinc, iron, copper, and oil shale mines in the United States (50), although few, if any, significant changes were made in the method until the mid-1960's. It was then, following the work of Cook (26) in the Republic of South Africa, that the Bureau began research with three-dimensional underground seismometer arrays for seismic source location. By this time, the quality and capabilities of available equipment had improved to the point where multiple-channel microseismic monitoring was possible and data could be recorded on magnetic tape. With these improvements came development of a broadband microseismic system capable of recording the true waveform of an event for detailed analysis (51) and multiple-channel systems to permit mine-wide determination of noise source locations (52).

Rock burst research in the Coeur d'Alene District resumed with the application of noise source location techniques that were applied to the problem of pinpointing zones of high stress concentration and, consequently, zones where a seismic event would most probably originate (53). The technique was tested in 1968 at the Galena Mine by locating the sources of microseismic activity during a 52-day period and preparing the cumulative noise source location plots shown in figure 11. These plots establish the noise sources, delineate the high stress concentration zones by the contoured areas of dense microseismic activity, and also indicate the sources of rock bursts that eventually occurred in the study area.

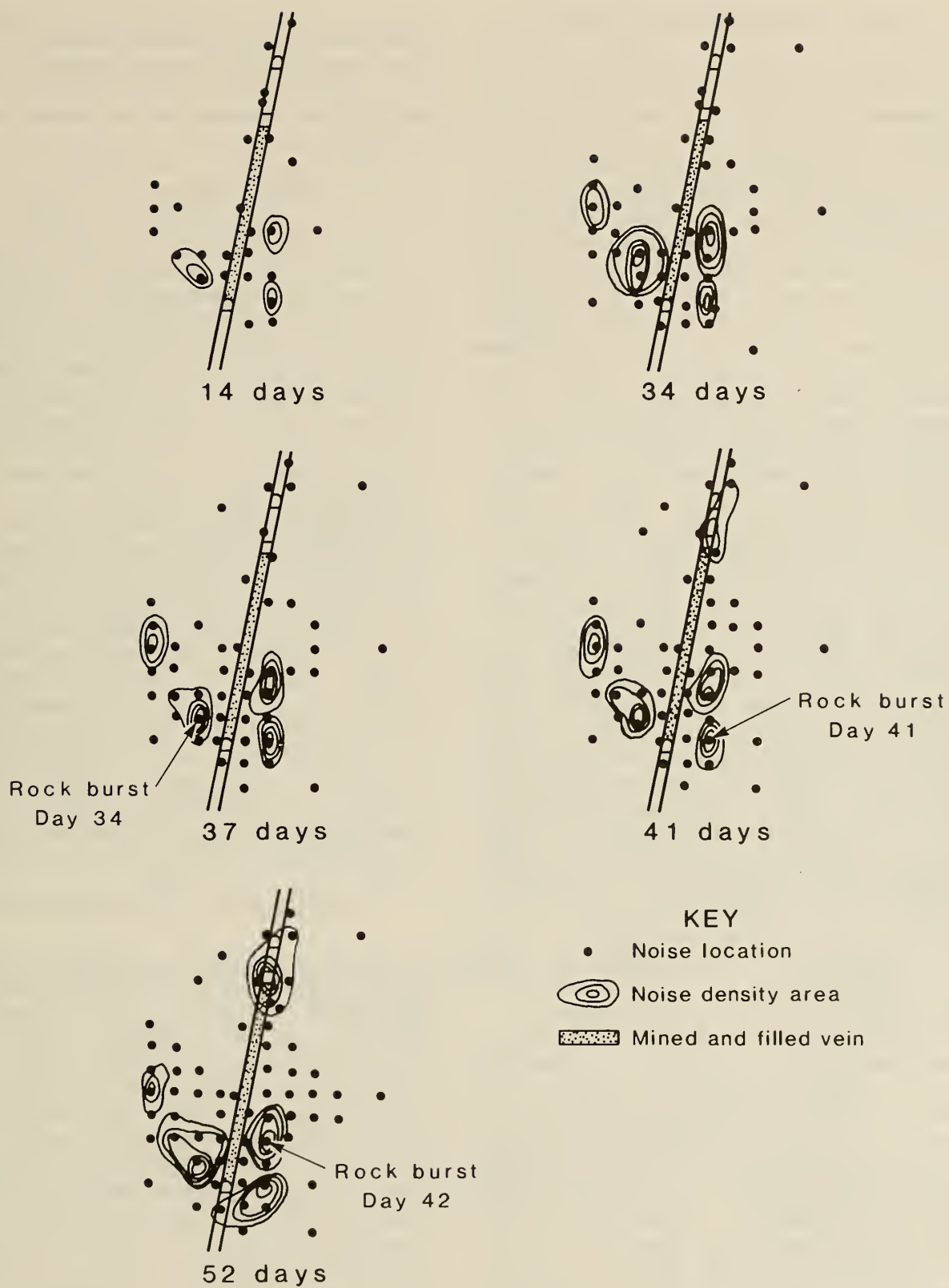


FIGURE 11.—Cumulative plots of noise source locations.

Rock Burst Monitoring Systems

Continued research by the Bureau in the early 1970's led to the development of a microseismic rock burst monitoring (RBM) system, followed by development of another monitoring system that uses a minicomputer (54). The original RBM system consists of an array of geophones with preamplifiers, high-gain amplifiers, the RBM control unit, and a printer. During operation, if a seismic event of greater than a preset threshold magnitude is detected by four or more geophones within a 0.2-s interval, the first arrival times of the signals are recorded, the relative energy of the signals is calculated, and the data are printed. The event location is then calculated from the first arrival times. The minicomputer RBM system also uses an automated data-gathering system but includes a minicomputer for automatic processing of the data, including calculation of the event coordinates.

One of the first district mines to install its own minicomputer RBM system was the Lucky Friday Mine, in 1972. The system monitored up to 23 geophones strategically located throughout the mine, processed microseismic noise data, and plotted the noise location coordinates (55). The original RBM system installed at the Lucky Friday Mine has been replaced to take advantage of the more sophisticated equipment developed since the first system was installed. During the same period, other district mines such as the Sunshine, Galena, Crescent, and Star also installed RBM systems.

The information obtained from microseismic monitoring systems is routinely used by mine personnel to detect and locate areas of high stress concentrations, after which control measures such as blasting to relieve stress or different stoping sequences to reduce mining-induced stresses are implemented. That these techniques have been successful in controlling rock bursts is indicated by a 1973-75 study of district mines (56). The results showed an overall decrease in the ratio of damaging rock bursts to total bursts, where total

bursts included those bursts triggered by destress blasting. The number of damaging rock bursts remained fairly constant over the study period, although the mines were being developed to deeper and more burst-prone levels.

Rock Failure Mechanisms

In addition to the use of microseismic systems at mine sites for rock stability and rock burst monitoring, such systems have been used in the laboratory and field to study rock failure mechanisms. Laboratory research during the late 1960's used microseismic emissions to locate microfracturing in rock specimens under increasing load, which provided insight into microfracturing and failure plane development (57-58).

The detailed analysis of microseismic field data provided valuable information concerning the influence of geologic structures on rock bursting, as well as the actual failure mechanisms. In many Coeur d'Alene District mines, noise source locations occur in bands that correspond to known structural features such as faults or hard quartzite layers interbedded with softer argillites. As previously mentioned, such structural controls are important because they may exert a significant influence on the concentration of mining-induced stresses.

DESTRESSING AND PRECONDITIONING

Destress blasting and rock preconditioning are rock burst control techniques developed to reduce the potential for rock bursts by uniformly fracturing a large section of solid rock. While destressing is used to reduce stress concentrations in highly stressed pillars remaining after a block of vein is mined, rock preconditioning is applied to a block of vein prior to mining and subsequent buildup of induced stress. In both cases, fracturing is achieved by drilling blastholes into the vein, loading them with explosives, and firing. In practice, a series of long, closely spaced holes are drilled into the vein from a crosscut, drift, or stope. The actual

hole pattern will depend to some degree on the available access to the vein and the drilling equipment used, but the ideal is a uniform distribution of holes.

The object of both control techniques is to produce uniform fracturing of the vein being destressed or preconditioned. In the case of destressing, fracturing has the effect of relieving concentrated stress by allowing rock displacement on the newly formed fracture surfaces and, consequently, allowing the dissipation of stored strain energy under relatively controlled conditions. Stresses that were concentrated on the small area of the pillar are then redistributed over the much larger area of the surrounding filled vein.

Since rock preconditioning is used on a block of vein prior to mining, fracturing of the rock has the effect of reducing its elastic modulus, and therefore its stiffness, before any abrupt increase in mining-induced stress. The rock burst potential is reduced because when the block is mined the rock is able to yield as stresses increase, rather than remaining rigid until increasing stresses cause a violent failure.

Pillar Destressing

The pillar destressing technique was first used in the Coeur d'Alene District during the late 1960's at the Galena

Mine (31). The purpose was to test the hypothesis that in order to produce a rock burst in a pillar, the stiffness of the pillar rock must be greater than the stiffness of the wall rock that comprises the loading system. When this condition exists and failure occurs, the stored strain energy in the wall rock will continue to load the pillar following failure, and kinetic energy will be released, producing a violent rock burst.

Microseismic monitoring of a pillar at the Galena's 3700 level indicated development of a potential burst zone in an area that had recently experienced a burst. The stiffness of the pillar and wall rock were calculated from a numerical model, and the pillar stiffness was found to be over twice that of the adjacent wall rock. Under these conditions of increasing stress in a stiff pillar, the potential for another rock burst was considered great.

In order to test the stiffness hypothesis, the pillar was destressed to reduce both its stiffness and the level of stress. Pillar and destress holes are shown in figure 12. The effectiveness of the technique was evaluated by the difference in seismic velocities through the pillar taken before and after blasting. Since seismic velocity is proportional to the rock's effective elastic modulus and thus to its stiffness, a decrease in velocity would imply a

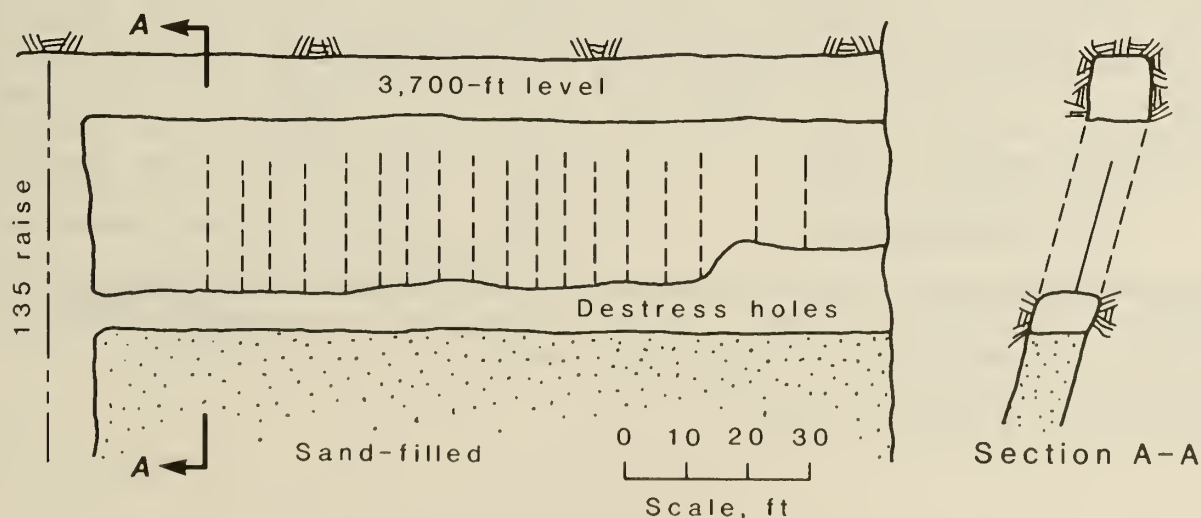


FIGURE 12.—Holes drilled into pillar from slope below for destress blasting.

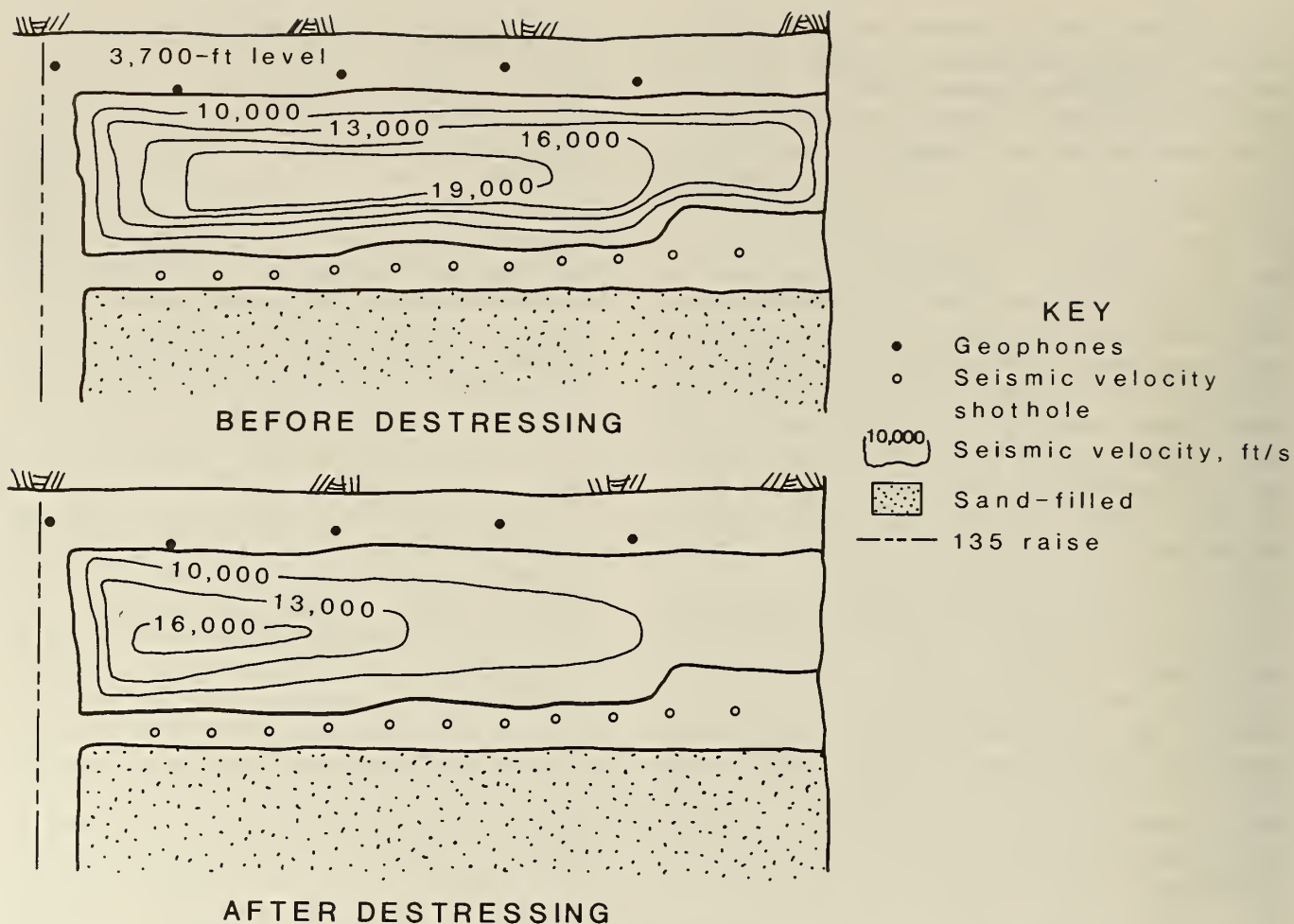


FIGURE 13.—Seismic velocity contours as measured before and after destress blasting.

reduction in stiffness. As shown by the velocity contours in figure 13, blasting did result in an overall decrease in seismic velocity. In addition, calculated pillar stiffness following blasting was less than that of the wall rock. The objective of destressing had been achieved, and the pillar was subsequently mined without further incident.

Rock Preconditioning

Based on the earlier success with the destressing technique, the idea of prefracturing or preconditioning a block of vein prior to mining was developed and demonstrated in a district mine.

First Demonstration

The first use of the rock preconditioning technique in the Coeur d'Alene District was in 1976 at the Hecla Mining Co.'s Star Mine (59). During development and mining of the 7500 level, unusually heavy rock bursting was encountered. After studying the stoping method being used, the stope geometry, geology, and stress conditions, Bureau researchers and mine personnel concluded that heavy rock bursting would also be encountered while mining the levels below the burst-prone section of the 7500 level.

The decision was made to test rock preconditioning by demonstrating its use on a small section of the 7700 level prior to mining. The block of vein selected, as shown in figure 14, was located directly below the area of heavy bursting encountered on the 7500 level and was accessible by the 7 and 10 crosscuts. Experience has shown that there are two times during the mining cycle when bursting is most probable: during opening of the initial or "I" drift at the bottom of a vein block and while mining the pillar produced at the top of the vein block. Preconditioning the vein 40 ft above and below the 7700 level would allow mining up from the 7700 level to begin in preconditioned rock and would also allow

later mining of the 7900 level to end in a preconditioned pillar.

Since mining of the initial drift from the 7 crosscut had already begun, the holes from this crosscut could not be drilled into the vein as were those from the 10 crosscut, but had to be located about 20 ft from the vein (fig. 15). The quartzite wall rock, not the vein, was preconditioned from the 7 crosscut. The blastholes were drilled and loaded from the crosscuts, as shown in figure 16, and detonated.

The preconditioning was monitored and evaluated by means of seismic velocity, stress, and displacement measurements. The reduction in seismic velocity following blasting indicated significant

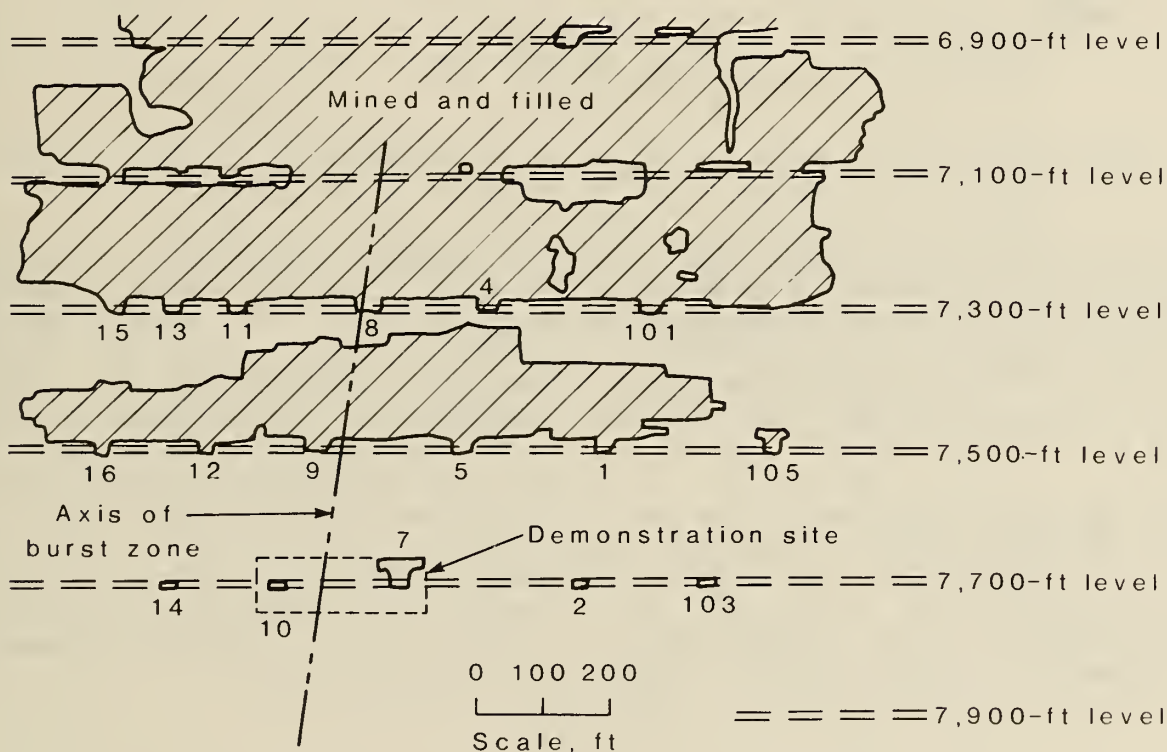


FIGURE 14.—Vein preconditioning, first demonstration site, 7,700-ft level of Star Mine. (Numbers identify crosscuts.)

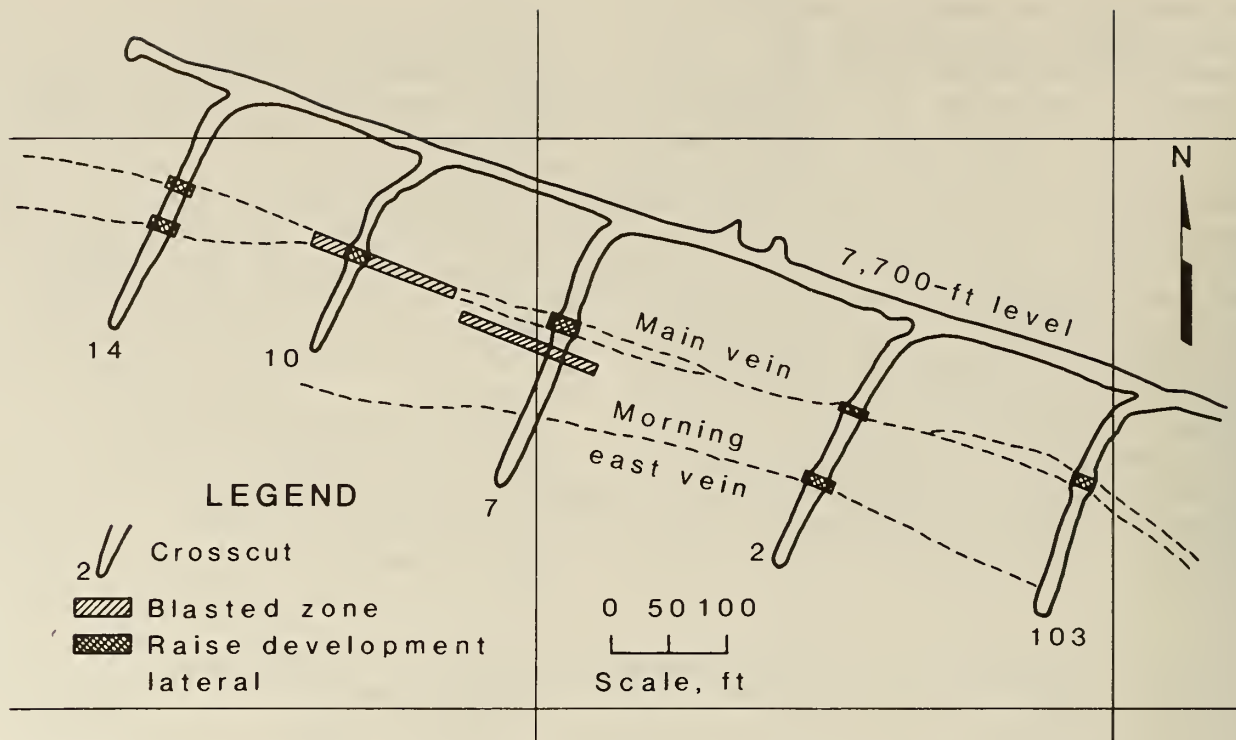


FIGURE 15.—Plan view of preconditioned zones, 7,700-ft level haulage lateral, and crosscuts to vein.

fracturing of the vein around the 10 crosscut and the wall rock around the 7 crosscut. Stress gauges, installed in the wall rock prior to blasting, showed a reduction in the horizontal component of stress, and displacement measurements showed an induced closure across the vein.

The real test came during subsequent mining. Since the wall rock, and not the vein, had been prefractured at the 7 crosscut, popping rock and typically difficult conditions were encountered when work resumed on the initial drift. Stopping of the first four floors, or 40 ft,

was accompanied by increased microseismic activity and one major rock burst. Stopping from the 10 crosscut, in prefractured vein, was without the usual rock popping or seismic activity. Even the initial drift was completed without problems, and during mining of the first 40 ft, microseismic activity was much lower than that from the 7 crosscut. During continued stopping of the 7700 level, above the influence of the preconditioned zone, seismic activity and rock bursting resumed, as was expected to happen without preconditioning.

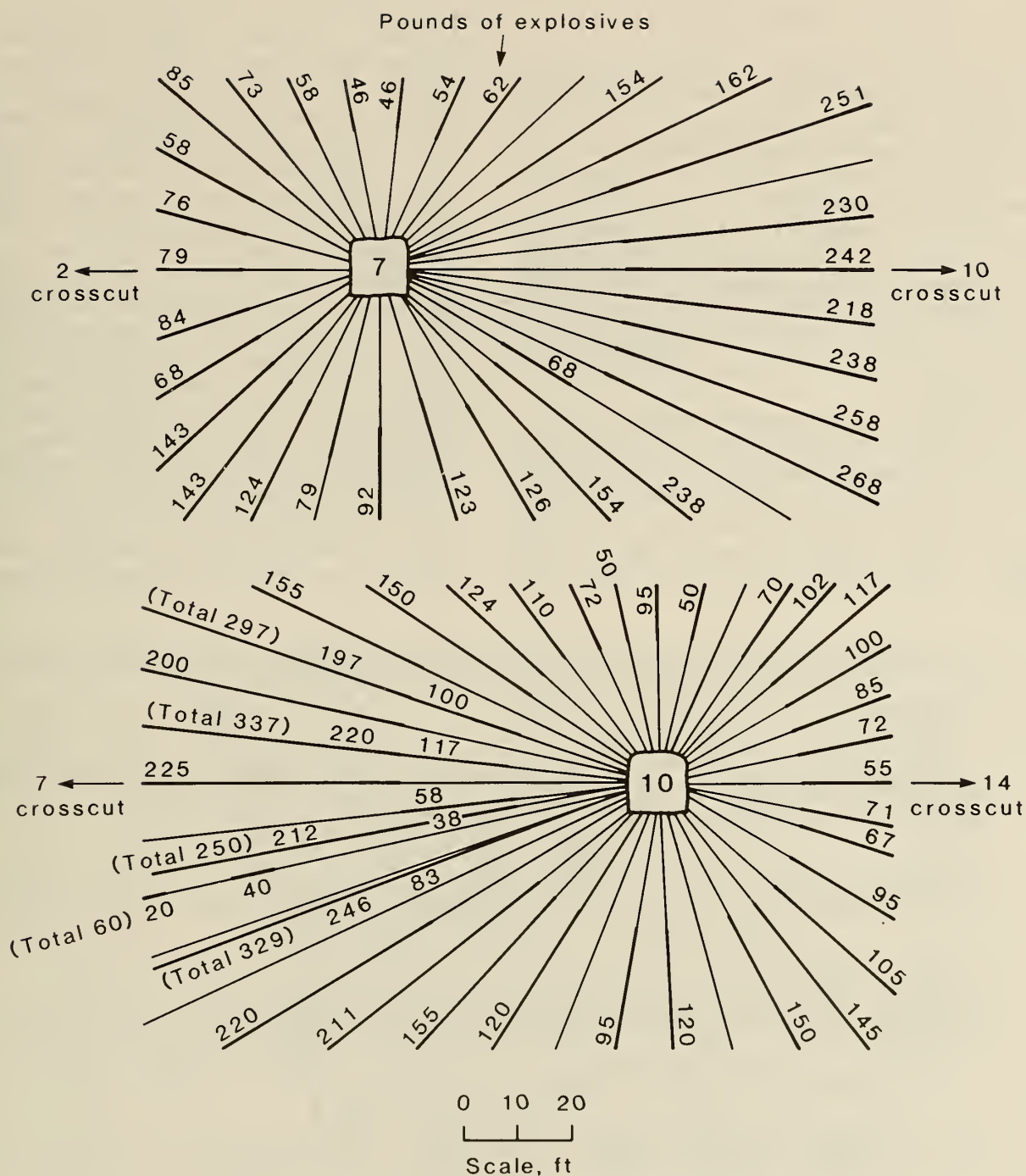


FIGURE 16.—Preconditioning drill-hole patterns and loading, 7 and 10 crosscuts, 7,700-ft level.

Second Demonstration

The experience gained by preconditioning a small section of the Star Mine's 7700 level and the reappearance of rock bursting above this section indicated the need to test the technique on an entire stope block. The location of this large-scale second demonstration was the Star Mine's 7900 level, as shown in figure 17. This area was chosen because it included the previously preconditioned zone above and was directly below the area of heavy bursting encountered on the 7500 and 7700 levels (60).

A primary safety concern about preconditioning the entire stope block was that prior horizontal stress on the 7900 level would be transferred by the preconditioning to pillars between the 7500 and 7700 levels. Any such increase in pillar stress would increase the pillars' rock

burst potential dramatically. To deal with this problem, preconditioning was divided into two phases separated by pillar destressing from the 7500 level and initial mining on the 7900 level.

The first phase, which was completed in 1979, consisted of preconditioning a narrow vein section from the 8 and 12 crosscuts on the 7900 level, while mining continued on the 7700 level above. The vein was drilled and loaded, as shown in figure 18, to produce a preconditioned area extending 50 ft above and below the level. Following preconditioning, mining from the 8 and 12 crosscuts began in the prefactured rock with little microseismic activity and no rock bursting.

Before the remaining stope block could be preconditioned in the second phase, the pillars between the 7500 and 7700 levels had to be destress blasted. Drilling was begun from the 9 crosscut on

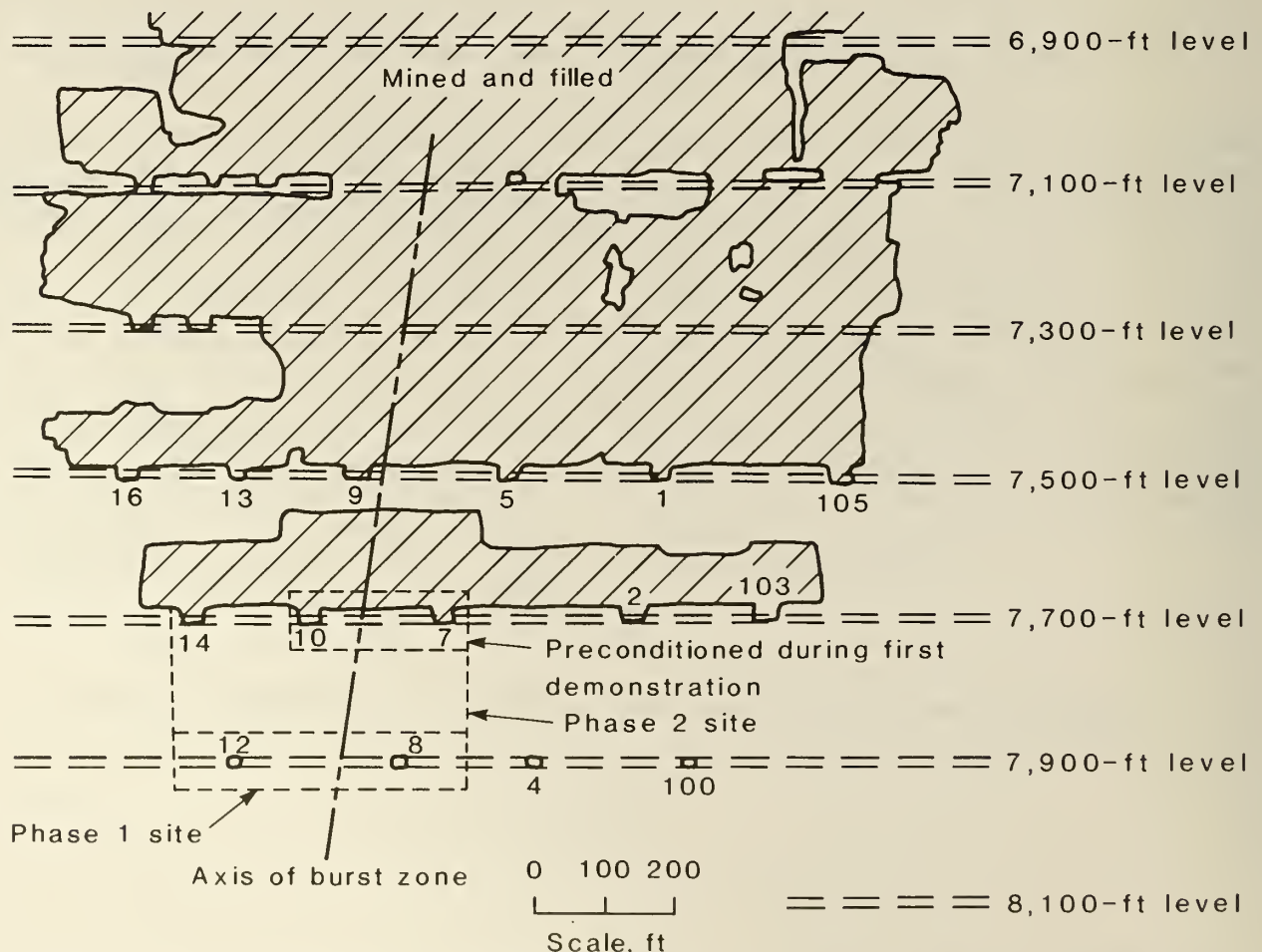
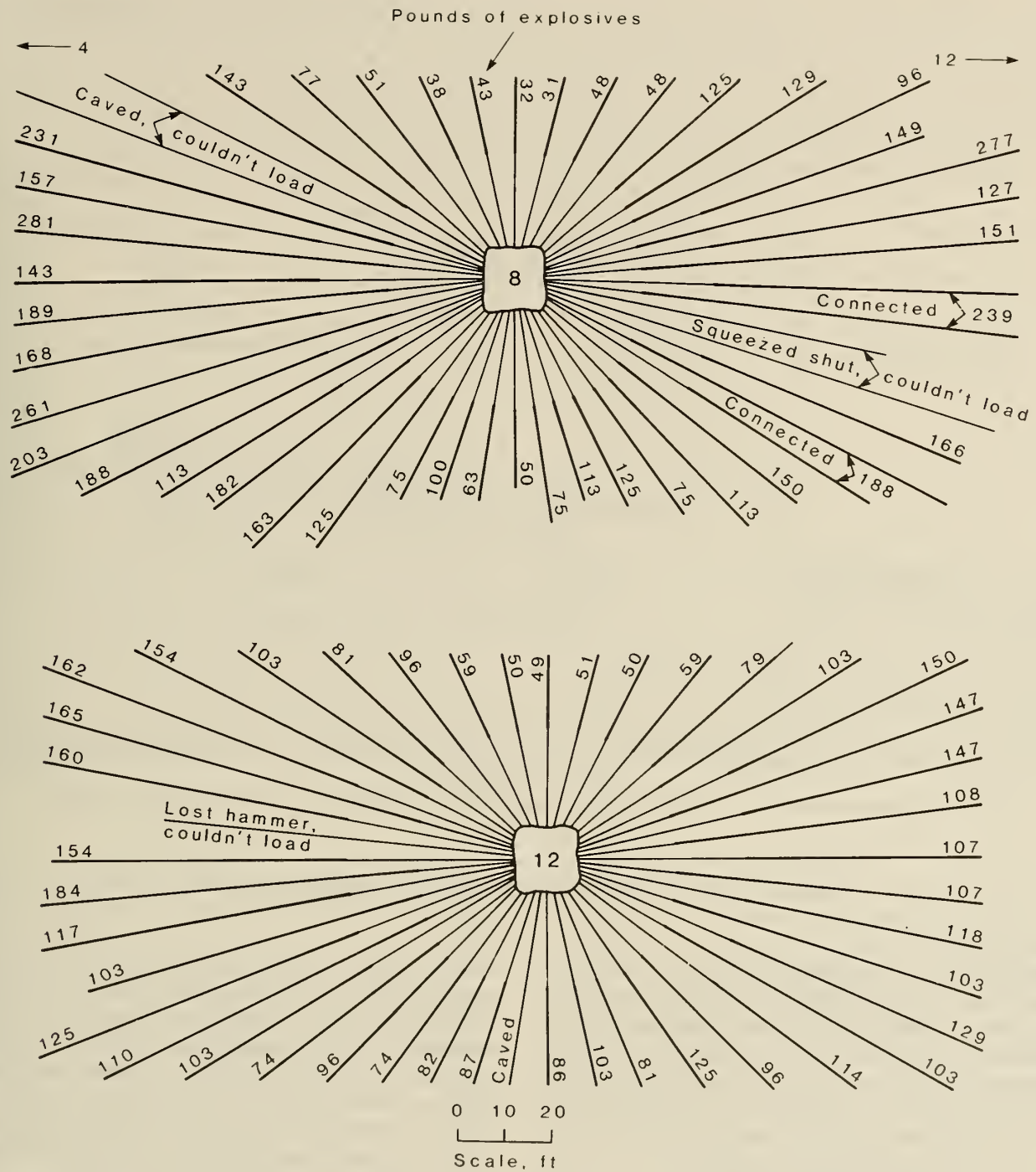


FIGURE 17.—Vein preconditioning, second demonstration site, 7,900-ft level of Star Mine. (Numbers identify crosscuts.)



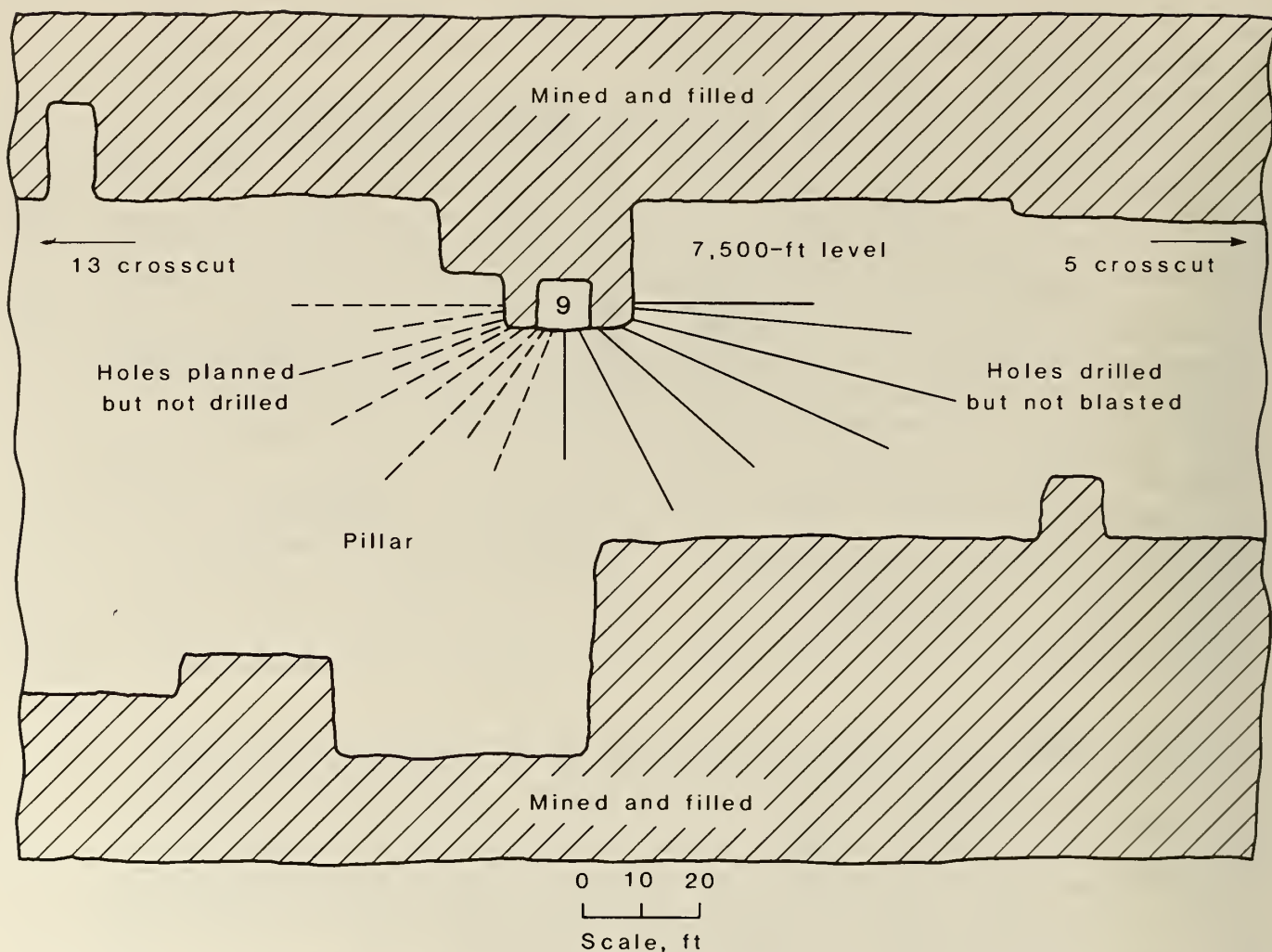


FIGURE 19.—Pillar destress blastholes drilled from 9 crosscut, 7,500-ft level.

7500 level, as shown in figure 19, but before drilling could be completed, a major rock burst occurred. The burst, of magnitude 2.6 on the Richter scale, effectively relieved stresses in the pillar below the 9 crosscut. The pillar below the adjacent 5 crosscut was drilled and blasted without any problems.

The second phase was completed in 1980 and consisted of preconditioning the remaining stope block by drilling vertically from the by-then-completed second floor of the 8 and 12 stopes. The drill holes are shown in figure 20 with the first phase area on the 7900 level. Following blasting of the stope, mining resumed with only moderate microseismic activity and no rock bursting until near

the top of the 8 stope. Microseismic activity in this area increased until a rock burst of 1.9 on the Richter scale occurred in a vein that had not been adequately prefractured. The vein was further destressed by drilling down from the 7 crosscut above and blasting (8).

While destressing and preconditioning have been shown to be effective rock burst control measures, their use involves some risk and high costs. When a block of vein is fractured by either method, some of the stress on the block becomes redistributed in the adjacent intact rock. This increases the level of stress on the intact rock and consequently increases the probability of a seismic event and resultant rock burst.

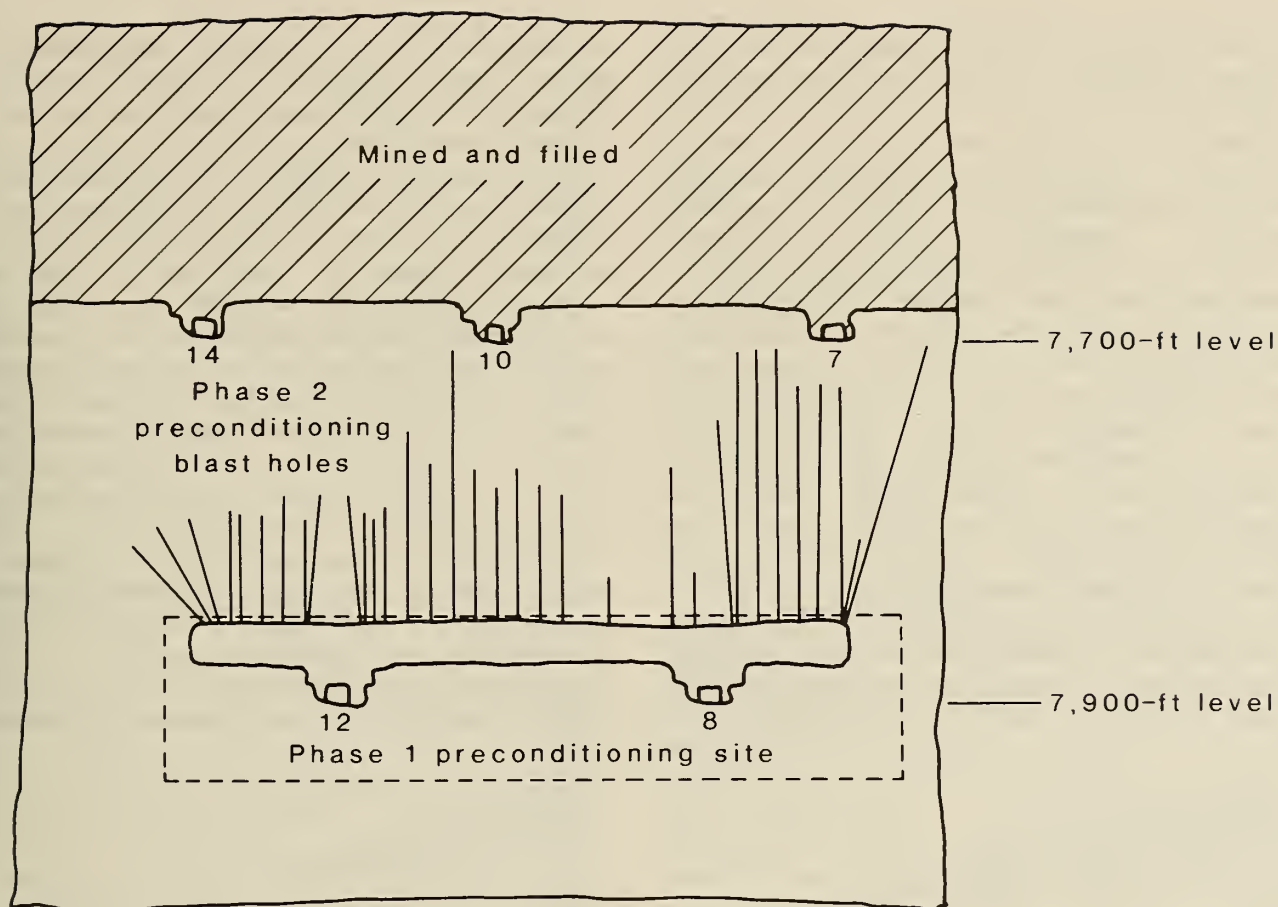


FIGURE 20.—Preconditioning hole pattern, drilled from 7,900-ft-level stope. (Numbers identify crosscuts.)

The techniques must therefore be used with careful planning and very close monitoring.

The major disadvantage of these stress control techniques is their high cost. Both techniques are labor- and equipment-intensive since a great deal of blasthole drilling must be done. To this, the cost of explosives must be added. These expenses only add to the already high costs of mining each ton of ore.

STOPPING METHODS AND MINE GEOMETRY

The influence that a particular mining method will have on rock bursting at depth is seldom a consideration during the initial planning and development of a mine. Historically, underground mining choices have been based upon ore deposit

and host rock properties, geologic structure, mine production requirements, and economic factors. The relationship between a mining method and rock bursting potential does not become a problem until bursting becomes a problem. By that time, unfortunately, the mine has usually been developed to a considerable depth and the mining method well established and difficult to change or modify.

Overhand Stopping

Coeur d'Alene District mines have commonly used the overhand cut-and-fill stopping method. This method is typically used to mine steep, narrow, high-grade vein deposits in strong, competent wall rock. The vein is divided into levels, usually 200 ft apart vertically, and each

level is mined from bottom to top, hence the name overhand. A vertical section of several mined and filled levels is shown in figure 17.

Access to the vein on each level is by crosscuts driven from a lateral haulage drift paralleling the vein, as illustrated in figure 15. The crosscuts, located about 200 ft apart horizontally, divide the vein into blocks. Mining proceeds from the crosscuts by "raising up" (excavating upward) into the vein and then excavating a series of horizontal stopes. Mining is done on a drill, blast, and muck cycle, and as each stope is mined out, it is backfilled with sand or waste material. Timber supports are added in some areas. The fill material provides support for the wall rock as well as a surface from which to continue mining above. Figure 21A shows a typical overhand stope with its raise and ore chutes, backfill, and active stope.

As each stope is mined and filled, the horizontal stresses become redistributed in the remaining intact vein above and below. Initial wall-rock convergence compacts the fill, and resistance of the fill to further compaction provides wall-rock support. While the fill modulus is normally much less than the wall-rock modulus, possibly as little as one one-hundredth, the support provided has been shown to be significant in terms of the stress induced in the remaining vein (61). The fill then provides more than just passive wall-rock support and a surface from which to continue mining—it plays an active role in the control of mining-induced stresses.

Overhand stoping produces a pillar of vein rock as mining proceeds upward toward the filled level. Typical pillars are shown in figures 17 and 19. The redistribution of stress with continued mining results in an increased stress

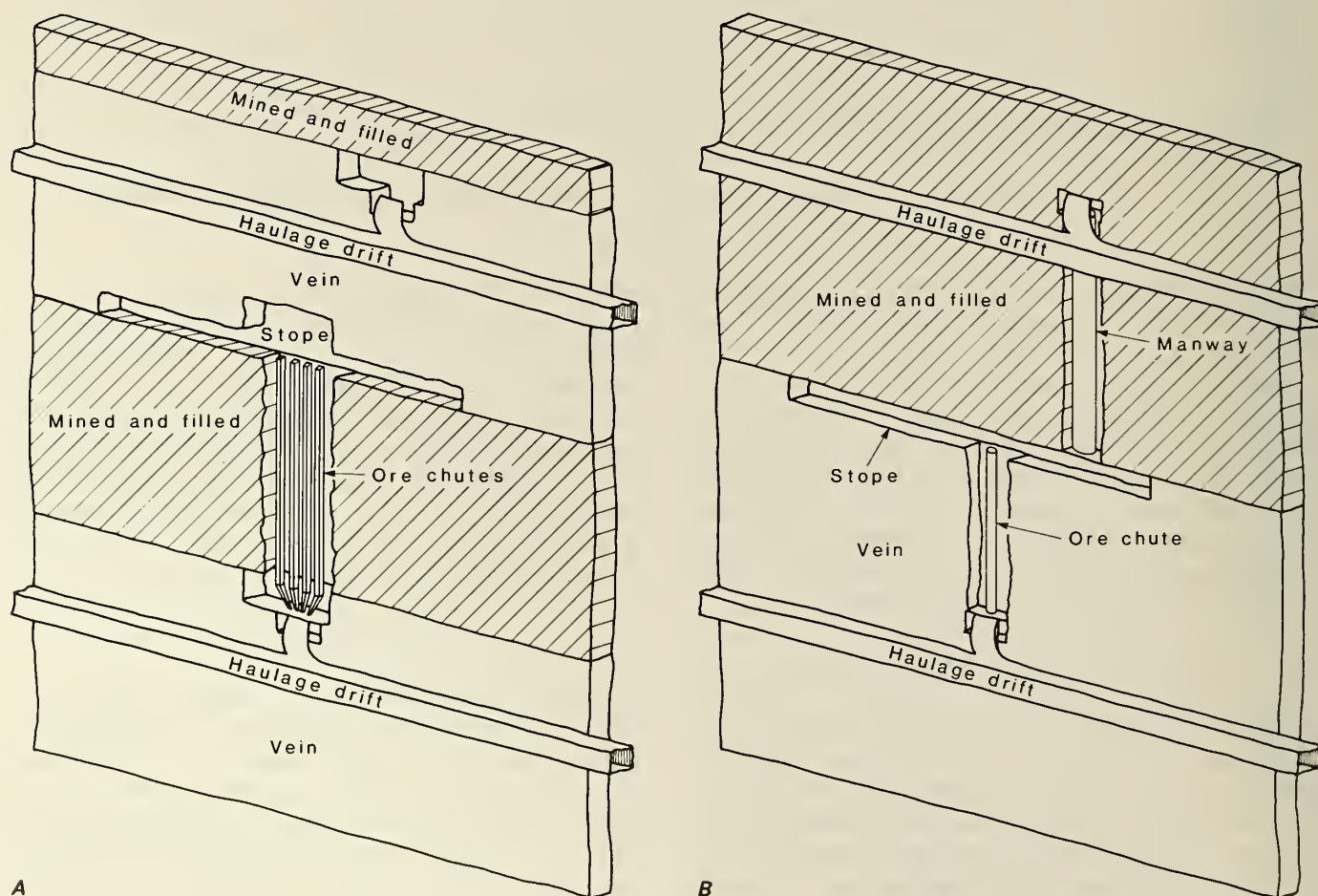


FIGURE 21.—Generalized drawings of (A) overhand and (B) underhand stoping methods.

concentration in the pillar until relieved by pillar failure. The nature of the pillar failure will be violent or nonviolent depending upon the relative stiffnesses of the pillar and wall rock. A nonviolent failure is characterized by gradual fracturing and yielding, while violent failure is often characterized by the release of seismic energy and a rock burst. Pillar bursts in overhand stopes account for many of the rock bursts experienced in the Coeur d'Alene District and are often responsible for severe damages since they are located directly above the active stope.

As district mines have been developed to greater depths and, consequently, into more highly stressed ground, rock bursting has increased in both incidence and severity. One approach to the alleviation of this problem has been the investigation of alternative stoping methods that minimize or reduce induced stress concentrations. Research by the Bureau has led to the development and testing of underhand cut-and-fill stoping methods as a means of reducing the rock burst hazard by eliminating the pillar in highly stressed narrow veins.

Underhand Stoping

Underhand stoping methods typically consist of mining a block of ore by cutting and filling in sequence from the top of the block to the bottom. A generalized drawing of a section of vein being mined by the underhand method is shown in figure 21B. Mining is done on a drill, blast, and muck cycle as with the overhand method, but downward since the intact vein is below the stope level. Following each cut, the open stope is backfilled with a cement-stabilized fill material. A cemented fill is necessary to prevent the fill from caving into the stope below when it is opened. Additional support and reinforcement techniques such as wire mesh, timbers, and rock bolts may also be required to stabilize the lower portion of each fill layer.

The primary advantage of the underhand stoping method over the overhand method with respect to rock bursting is the absence of a pillar. Since the mining direction is downward into a continuous vein, a highly stressed pillar is not formed. Instead, abutment stresses remain in the intact vein and wall rock immediately below the open stope and move downward as mining proceeds downward. Above the open stope, the cemented fill is compacted by wall-rock convergence, and displacement of the wall rock produces a region of reduced stress.

If, for some reason, the underhand method is simultaneously used on more than one level, sill pillars will be formed as mining approaches the bottom of each level. These pillars will then undergo the same increasing stress concentrations as pillars formed by the overhand method. A model for simulating various overhand and underhand stoping sequences is shown in figure 22 along with the order of elements mined. The calculated energy release rates for these simulations are shown in figure 23A and B. Figure 23A compares energy release rates of overhand and underhand stoping methods, and illustrates the abrupt increase in energy release rate as the overhand pillar is mined. The underhand method results in a uniform rate of energy release. Figure 23B compares energy release rates for single- and multiple-level underhand stoping sequences. The energy release rates produced by multiple-level underhand stoping approach those of overhand stoping when sill pillars are formed and mined (62).

While multiple-level underhand stoping presents an increased risk of rock bursting, the method still retains advantages over the overhand method. One advantage is that if pillar destressing or rock preconditioning techniques are applied to the vein, subsequent mining will proceed from above the prefractured rock and not from below it. This eliminates the hazard associated with working under fractured and possibly loose rock. Another

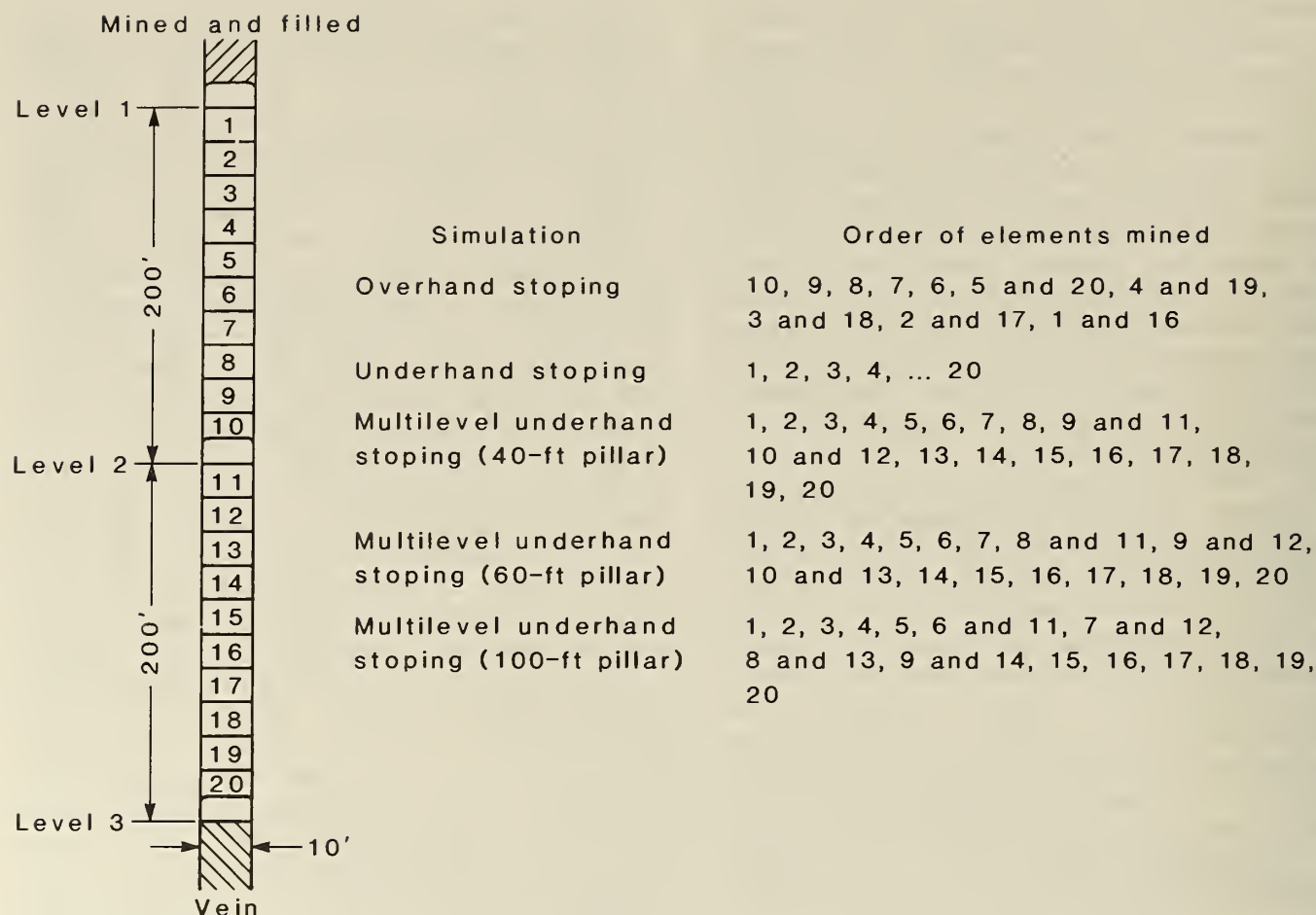


FIGURE 22.—Model for simulated overhand and underhand stoping of section of vein.

advantage is that if a rock burst should occur, it will be more apt to occur in the vein below the stope. The stope damage would probably be less severe, with safer cleanup and repair work, than if a rock burst above caused caving into the stope.

Underhand Stoping Demonstration

Initial research into underhand stoping methods as a means to reduce the rock burst potential in Coeur d'Alene District mines was conducted in two phases. During the first phase, several mines in the

United States and Canada that use the method were examined and studied to determine current practices. Feasibility and cost-effectiveness assessments were made, and based upon the study results, initial design recommendations were prepared.

The second phase of research consisted of a demonstration test of underhand cut-and-fill mining in a district mine. The demonstration site had to have a history of rock bursting, include pillar distress blasting, and be in an active mining location so that a cost comparison could be made.

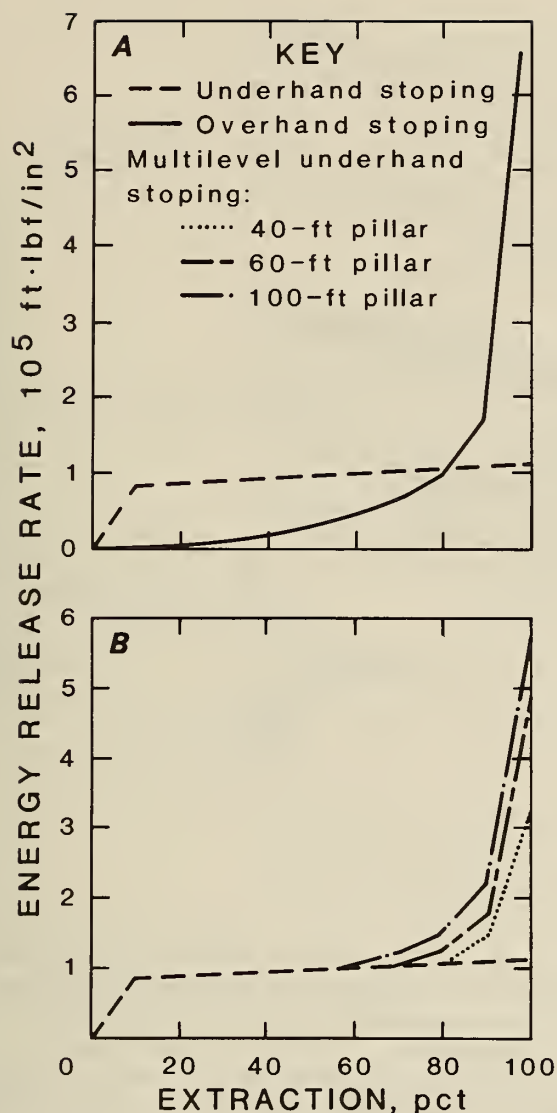


FIGURE 23.—Energy release rates for simulated stoping. A, Overhand and underhand stoping; B, multilevel underhand stoping.

The site chosen was the 6700 to 6900 level of the Grouse vein at the Hecla Mining Co.'s Star Mine. This portion of the vein, shown in figure 24, had been mined by overhand stoping to within 50 ft of the 6700 level. A rock burst had occurred in the pillar near the 125

raise, and based on the vein's history, additional bursts were anticipated as the remaining pillar was mined.

The plan for the underhand stoping demonstration was to destress-blast the remaining pillar by drilling vertically from the stope below, backfill the 6700-level haulage drift, and mine the pillar from above by underhand cut-and-fill stoping.

To monitor the demonstration, instruments were installed in the 6700-level haulage drift. Included were extensometers for wall closure, stress gauges for wall-rock stress changes, and soil pressure cells for measuring loading on the cemented fill. Extensometers and pressure cells were added to each stope while that stope was open to continue monitoring closure and fill loading as the pillar was mined. The seismic velocity through the pillar was determined both before and after destress blasting to check the effectiveness of fracturing.

While instruments were being installed, the 6700-level haulage drift was prepared for the cemented fill by construction of a floormat to restrain any loose fill during subsequent mining below. The floormat construction details are shown in figure 25. The wire mesh and porous fabric were necessary to retain solids while draining water from the cemented fill. As part of the demonstration, an experimental cement plant was set up for preparing and pumping fill material from the surface to the stope.

Destress blastholes were drilled vertically into the pillar, except in the future raise areas shown in figure 26. These areas were not blasted because it was feared that fractured rock would cause problems when the raises were driven. Once the holes had been drilled and loaded and the 6700-level drift filled, the pillar was blasted.

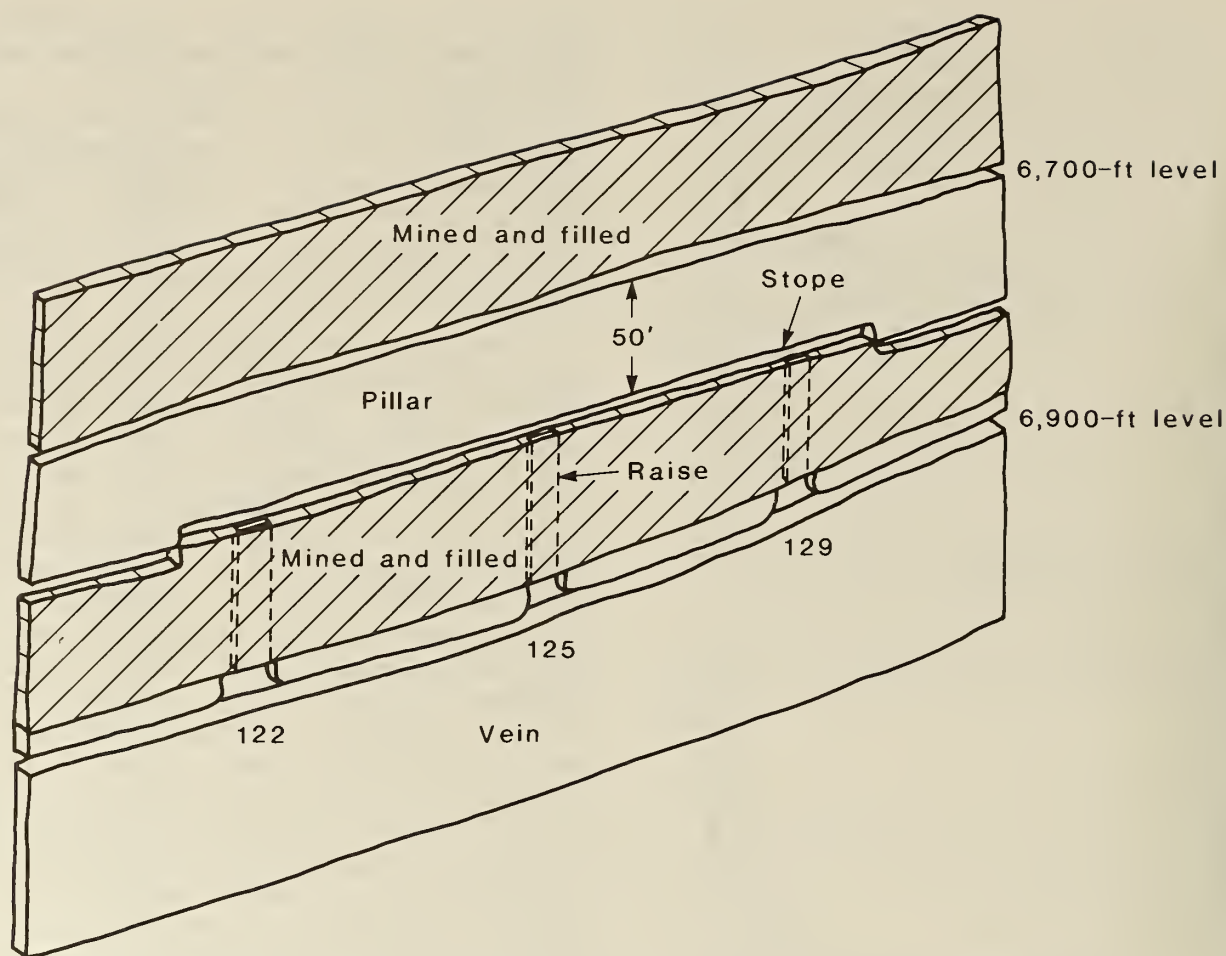


FIGURE 24.—Underhand stoping demonstration site, 6,700-ft level of Grouse vein, Star Mine. (Numbers identify crosscuts.)

Raise driving up through the pillar to the 6700 level began at the 122, 125, and 129 stopes following blasting. The 122-stope raise was driven without difficulty, but some problems were experienced with the 125- and 129-stope raises. The 125 raise passed through the region of rock fractured by a prior rock burst, and loose, broken rock became a problem while the raises were driven up through the pillar. Problems on the 129 raise were related to overloaded blasting rounds, which caused excessive fracturing of the already fractured rock.

Following completion of raise driving, the first underhand stoping cut was begun beneath the cemented fill of the 6700 level. As mining proceeded in each direction from each raise, two problems became evident. The first was that the quality of the cemented fill was inconsistent, and the second was that fill material in the drift above crushed and buckled because of the large amount of wall-rock closure.

The quality problems were the result of inconsistent cement and fill mixing and poor proportioning of the fill material,

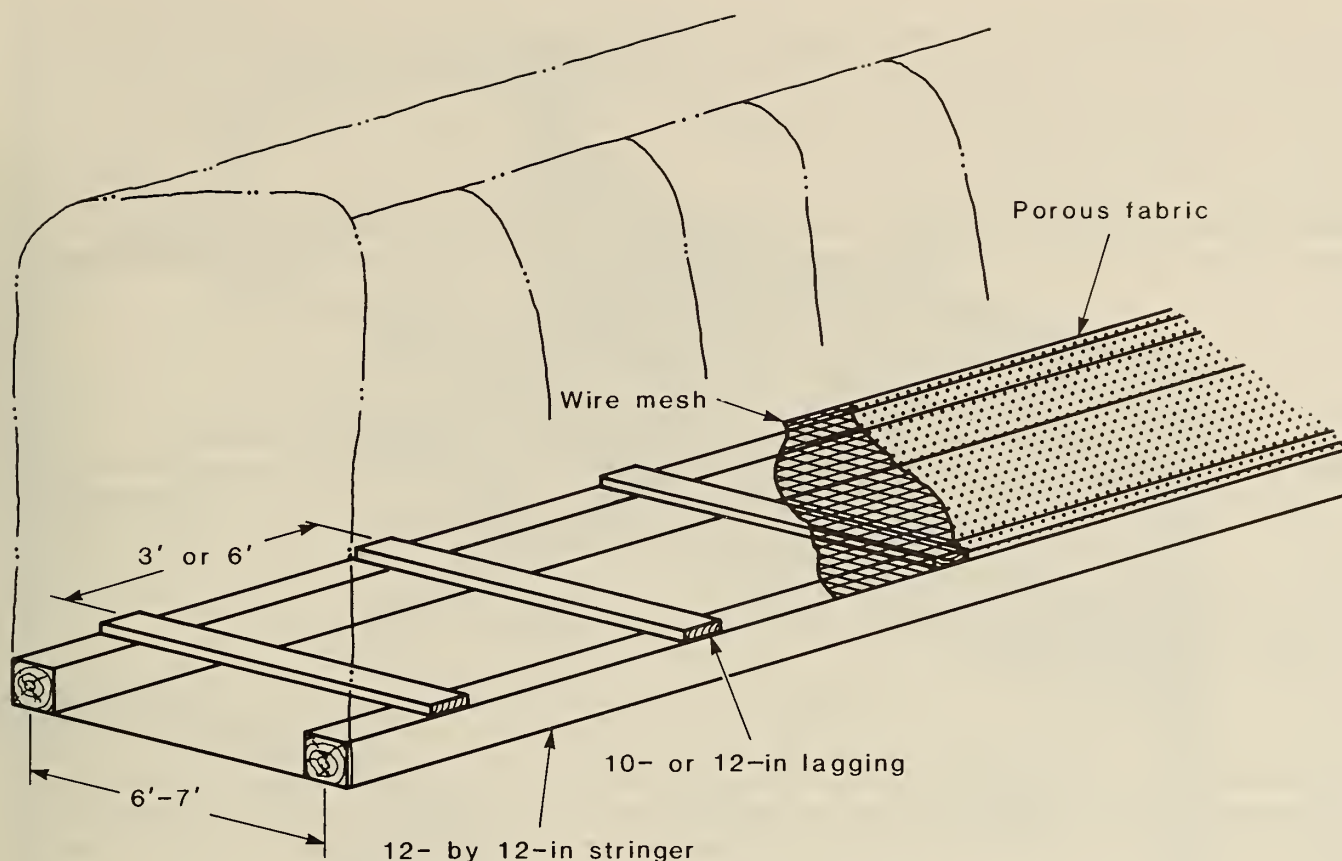


FIGURE 25.—Floormat construction details for cemented fill material.

which caused a separation and loss of cement from the fill during pouring. The cement and fill mixing problem was easily corrected at the surface mixing site, but the proportioning of fill material continued to be a problem. During the filling of later cuts, the problem was partially corrected by filling the first 3 or 4 ft and then allowing the cement to settle before adding the remaining fill.

The great amount of wall-rock closure following mining indicated that pillar distress blasting had been successful but caused problems in maintaining the raise openings. As a consequence, continued raise repair work was required while the pillar was mined. Extensometers in the

6700 level indicated about 4 in of closure with the first cut, but during the year required to mine the pillar, approximately 14 in of additional closure occurred in the raise areas.

Four underhand cuts were required to mine the pillar. The only problems encountered during mining involved overloaded blastholes and blastholes drilled too close to the filled stope above. These damaged the floormat and fill above, but were easily corrected with experience and more careful drilling and blasting procedures.

Overall, the project successfully demonstrated that underhand cut-and-fill stoping with massive distress blasting

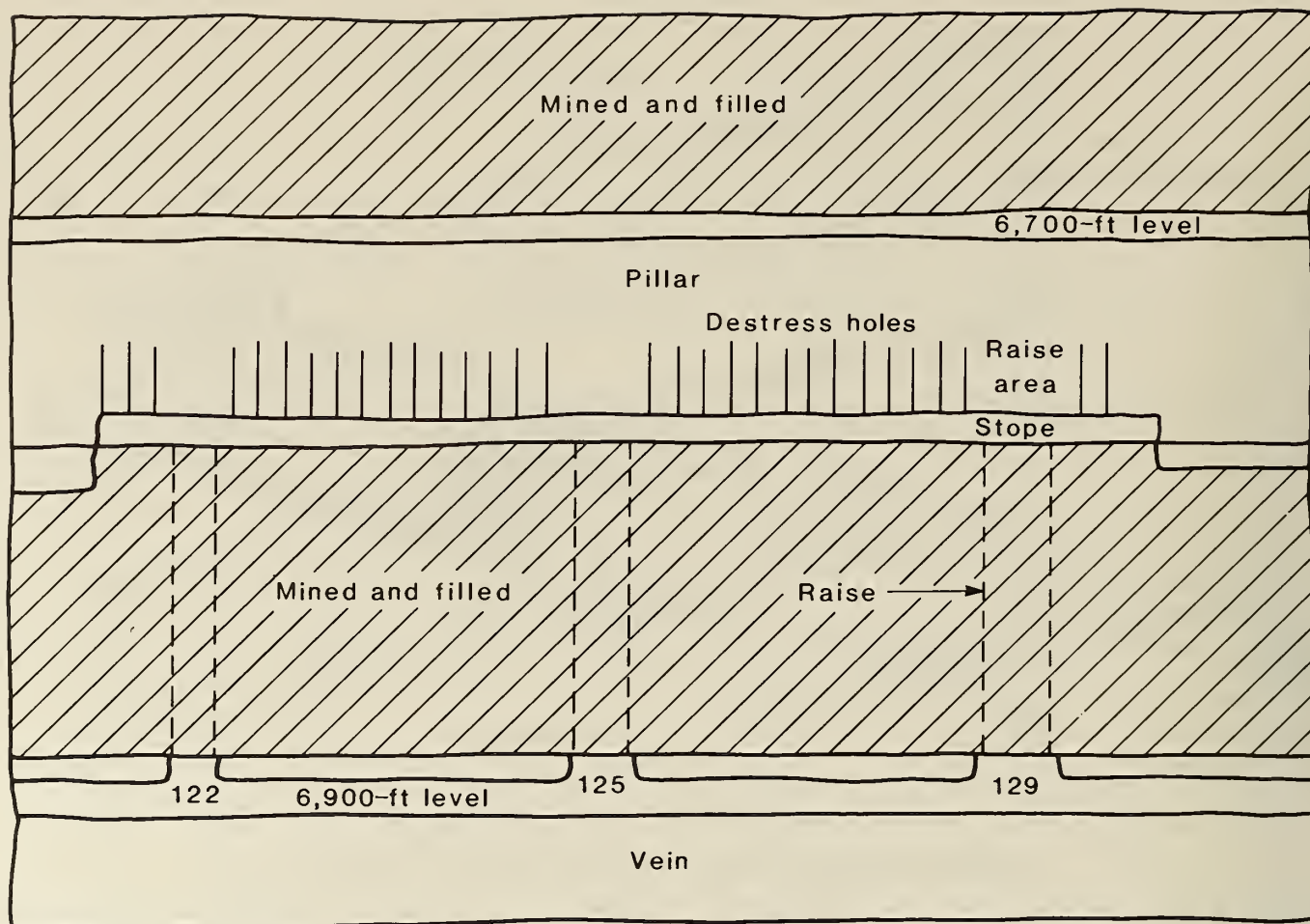


FIGURE 26.—Pillar destress blastholes drilled from stope below 6,700-ft level. (Numbers identify crosscuts.)

can be used in the Coeur d'Alene District mines to control rock bursts. Even though the pillar was highly burst prone, only one moderate burst took place during the project. That burst was located in the wall rock and occurred during the first cut. Damage from the burst was minor and limited to some loose rock being shaken down in the nearby stopes.

Lucky Friday Underhand Longwall

Conceptual planning for the second demonstration of underhand cut-and-fill mining in the Coeur d'Alene District began in 1983 at the Hecla Mining Co.'s Lucky Friday Mine. The threat of increased rock bursting with continued mining at greater depths prompted a decision to develop an inherently safer mining method. In addition to providing a

measure of control over rock bursting, the method would also have to provide the required production to be competitive.

A site for the Lucky Friday underhand longwall was set aside between the 5100 and 5300 levels and the 106 and 110 crosscuts (fig. 27). This site provides a 500-ft length of narrow, nearly vertical vein at a point where the vein turns from a north-south to a northeast-southwest strike. The vein block above the 106 crosscut would be mined by the conventional overhand cut-and-fill method while at the same time the vein below would be mined by the underhand method.

Initial plans considered the use of steel-lined mill hole raises in the vein as an ore pass from the stopes to the 5300 level. Development and testing of the design and the experience with raises at the Star Mine led to the conclusion

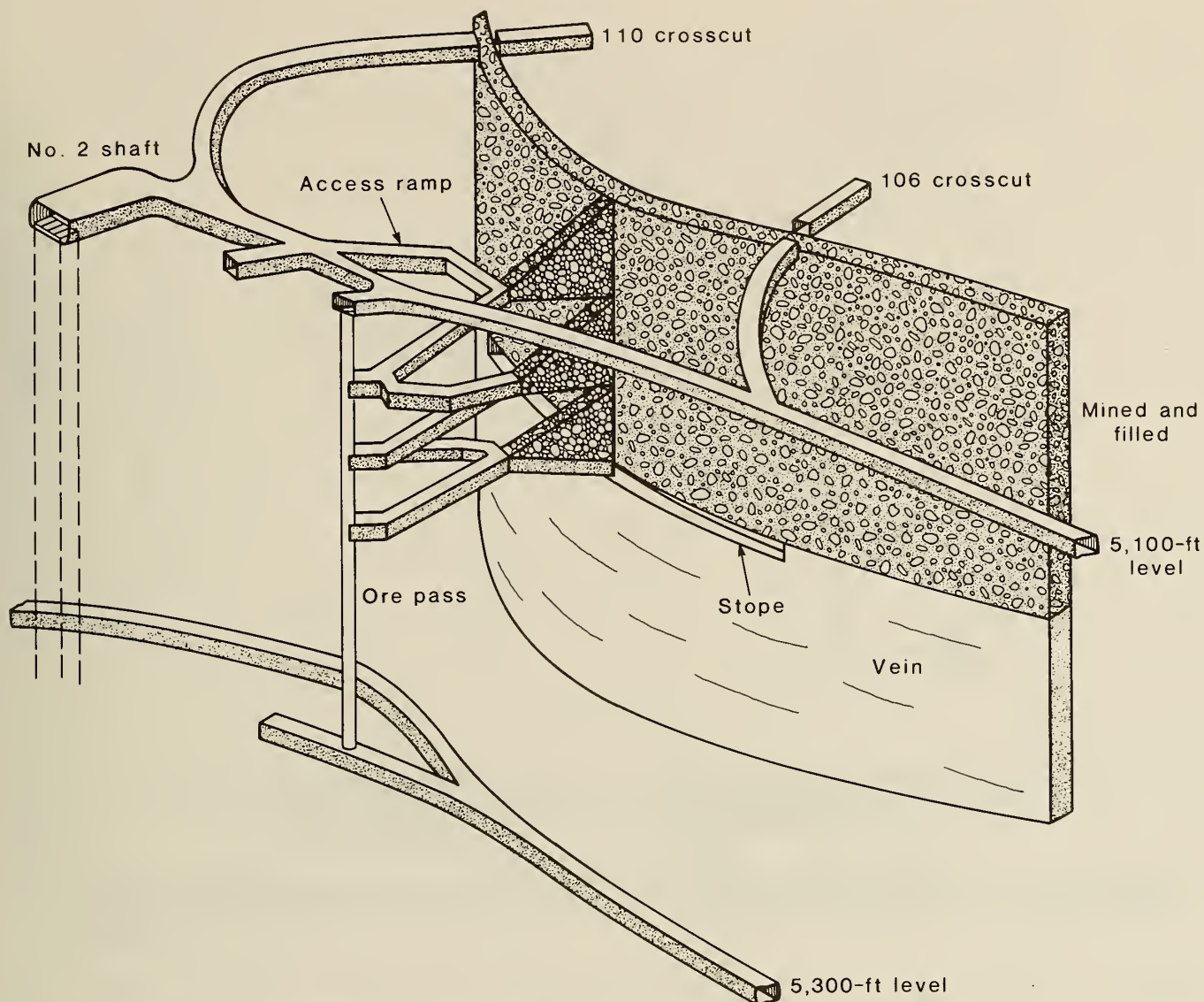


FIGURE 27.—Isometric view of Lucky Friday underhand longwall site.

that boring and maintaining openings in the vein would present very difficult problems. In the existing high horizontal stress field, rapid deformation of the hole following boring and reaming could make it impossible to maintain a circular opening. Once completed, the risk of wall caving or spalling prior to or during liner installation would be high and would present a serious safety hazard. Furthermore, during subsequent underhand stoping, the expected wall-rock closure would aggravate any existing problems, making maintenance very difficult.

Because of these potential problems with mill hole raises, a ramp system with

an ore pass distant from the vein was developed, as shown in figures 27 and 28. The ramp allowed placement of the ore pass away from the area of high induced stress at the vein and also permitted the use of diesel-powered load-haul-dump units. During underhand stoping, ramp development was kept just ahead of mining to provide for vein access and ore haulage. The ramp design consisted of alternating south, center, and north ramps, which spaced the openings farther apart for greater stability in the highly stressed wall rock.

A part of the project requiring much research and development was that of the cemented-fill system for backfilling the

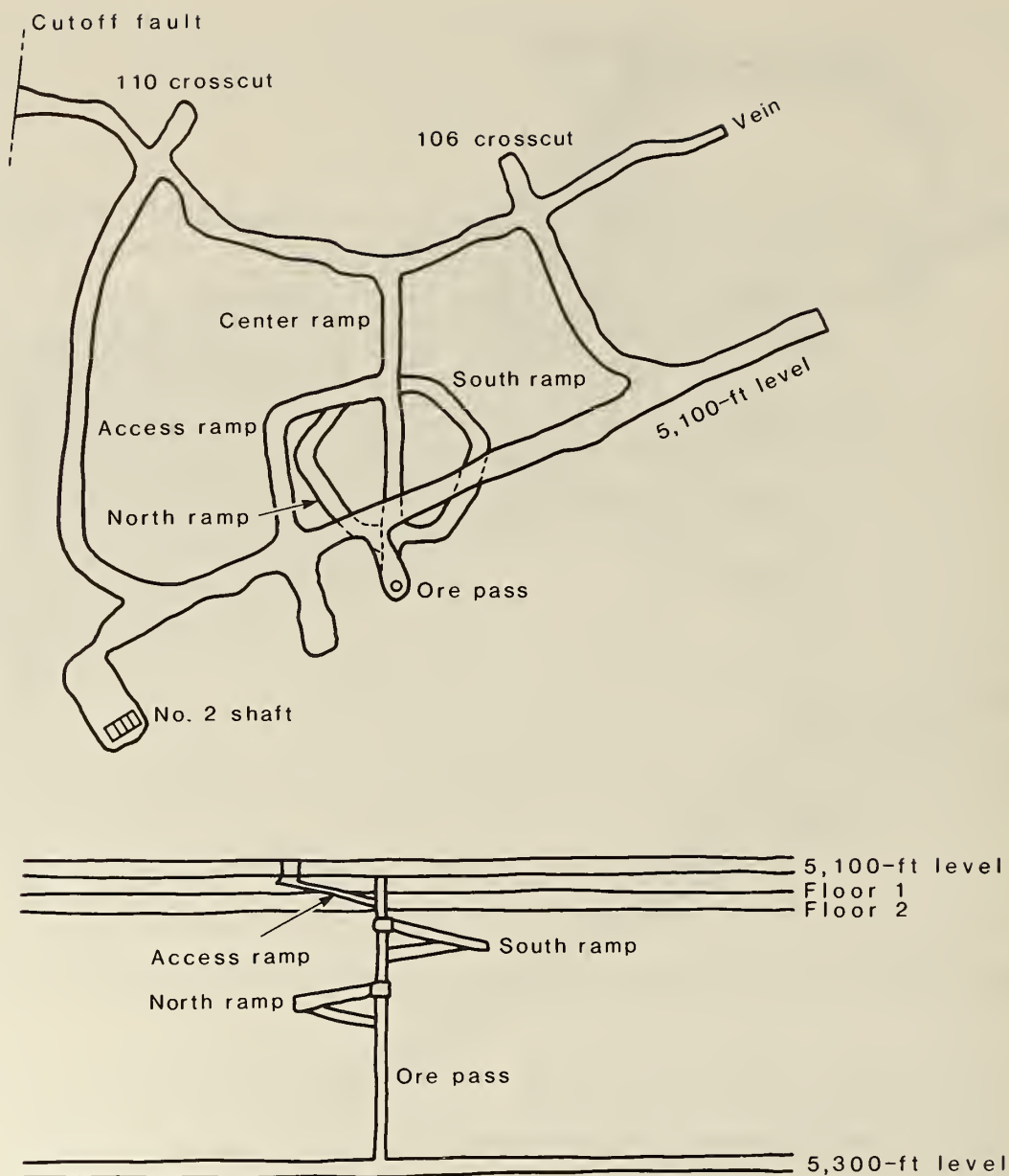


FIGURE 28.—Plan and cross section of Lucky Friday underground longwall site.

stopes. Initial research centered on methods of preparing a fill material of adequate strength to resist crushing by wall closure. Closely coupled with the problem of preparation was the problem of transporting the fill mixture underground, since the transport system influences the equipment required for mixing. The fill system developed mixed mill tailings with cement in a concrete batch plant at the surface and then used a pump and a pipeline to transport the cemented fill to the stopes.

A two-dimensional, finite-element model of a vertical section through the vein and ramp system was prepared to simulate both overhand stoping from the 5100 to the 4900 level and simultaneous underhand stoping from the 5100 to the 5300 level. The model simulated cut-and-fill stoping with sandfill for overhand stoping, cemented fill for underhand stoping, and development of the ramp haulage system. Analyses of the modeling results show the development of induced stresses

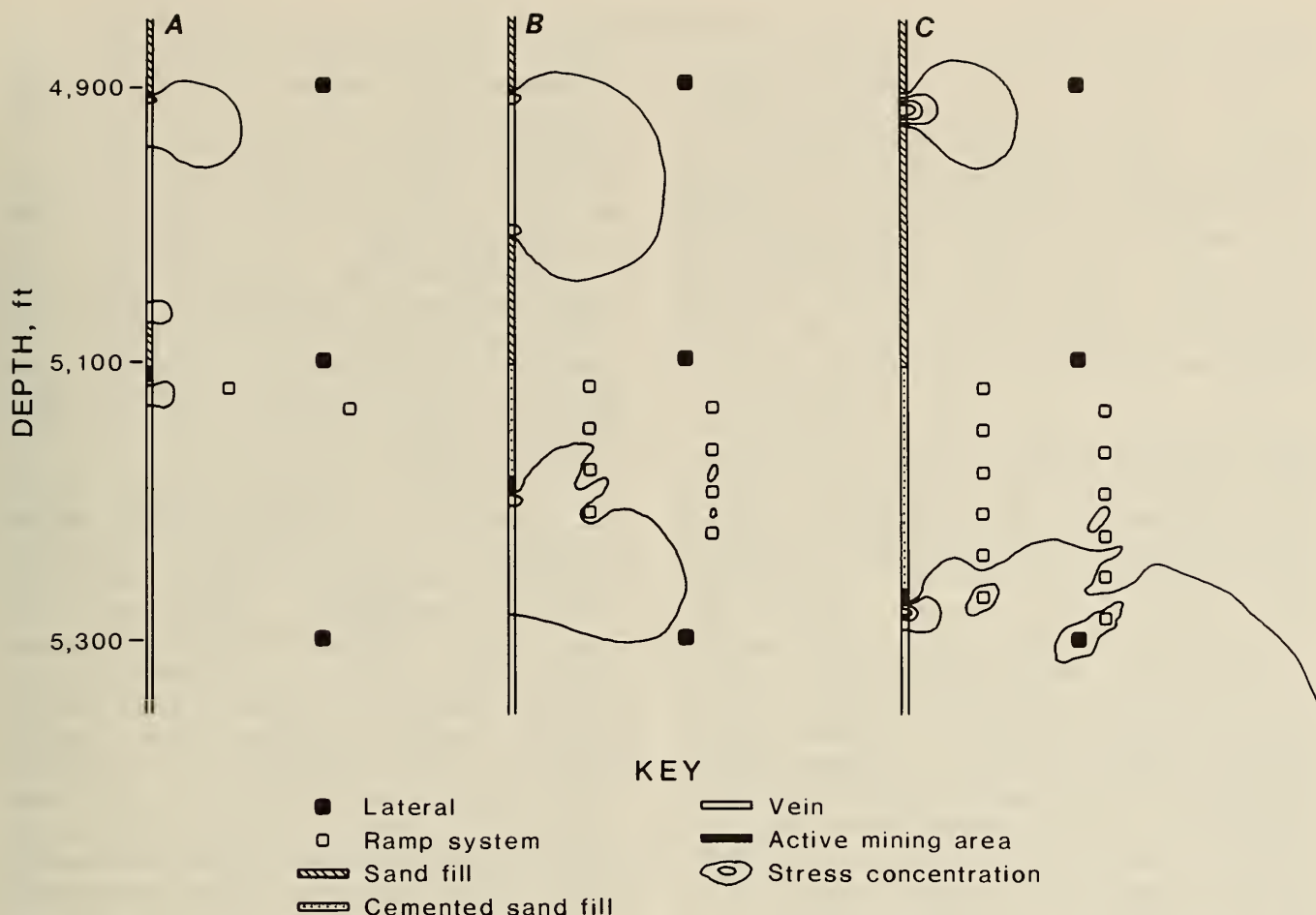


FIGURE 29.—Mining-induced stress concentrations from simulated overhand (above 5,100-ft level) and underhand (below 5,100-ft level) stoping. A, Start of stoping; B, half of each level completed; C, stoping nearly complete.

with each method. The stress contours shown in figure 29A, B, and C illustrate the concentration of stress in the pillar as overhand stoping approaches the 4900 level. Stresses induced by underhand stoping generally remained in the vein and wall rock below the active stope, while the ramp area remained clear of the high induced stresses near the vein.

As in prior demonstrations, instruments were installed to monitor stresses and stope closure during mining. These instruments included extensometers in the ramp and stope openings and pressure cells in the filled stopes to monitor wall loading on the fill. In situ stress measurements were also taken, and all field data were used to verify the

finite-element model as well as monitor the response to mining.

Development of the ramp system to approach the vein began in early 1985 and was followed by the start of underhand stoping at the 5100 level. When the first stope had been completed, it was backfilled with cemented fill, and the second underhand stope was begun below the fill. Unfortunately, weak market conditions forced suspension of operations at the Lucky Friday Mine before the second stope could be completed. Although some minor rock bursting was encountered during development of the ramp system, no major design problems have been found in the overall mining plan.

CONCLUSIONS

The Coeur d'Alene District of northern Idaho is one of those mining districts where a unique combination of high in situ stresses and hard, brittle rock produces violent rock failures known as rock bursts. In the Coeur d'Alene District, as in many other districts around the world, rock bursting did not become a serious operational problem until the mines had been developed deep into the rock.

Rock bursts are violent rock failures that release seismic energy. When this radiated energy encounters a mine opening or is released at the face of an opening, the resulting rock burst may be capable of destroying structures or mining equipment and causing severe injuries and fatalities. Based on experience and the results of extensive studies, certain conditions are necessary in order for a seismic event to occur and release stored strain energy: the in situ and mining-induced stresses must strain some point or region in the rock to a condition of unstable equilibrium, and then an additional stress must trigger failure. The failure, once initiated, must be a violent, brittle type of fracturing, and there must be available stored strain energy released as kinetic or seismic energy. The actual rock failure

mechanisms that release seismic energy are only poorly understood at best, even after many years of intense research in both field and laboratory. Several important failure criteria have been postulated to explain the conditions for the initiation of failure, and more recent work has attempted to explain the postfailure behavior of rock. The development of rock behavior simulation by numerical modeling will provide a very useful means for calculating the probability of rock bursting.

Bureau of Mines research in rock bursts has focused on development of microseismic monitoring and analysis techniques, destress blasting and rock preconditioning, and alternative stoping methods to reduce burst hazards. The microseismic method has become a refined tool for locating seismic events and gaining better understanding of rock failure mechanisms. In the Coeur d'Alene Mining District, destress blasting and rock preconditioning methods have been demonstrated to be effective means of working a burst-prone mine area. After initial field demonstration projects, underhand stoping methods are being developed further to permit safer mining in highly burst-prone, deep vein mines.

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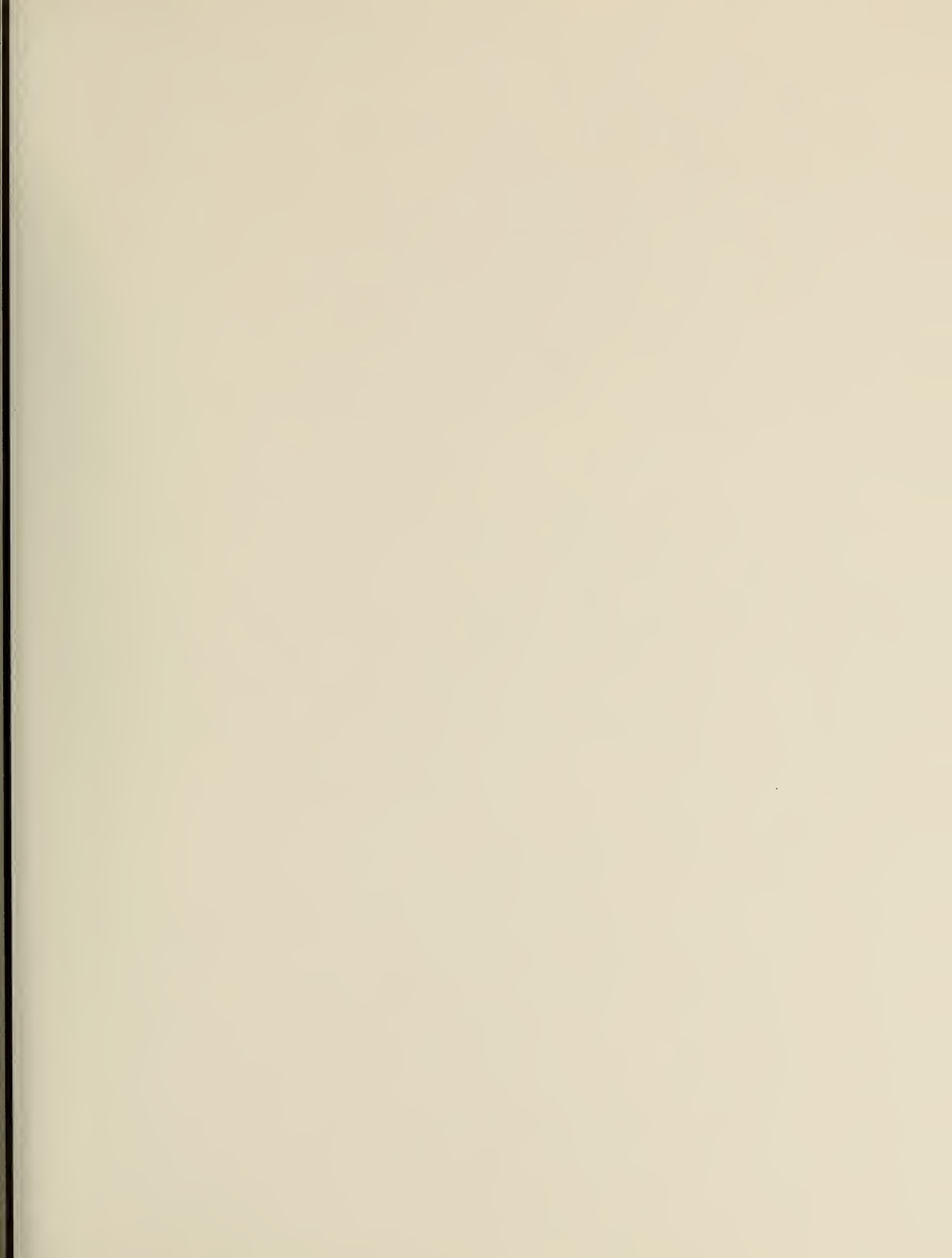
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